NI 43-101 Technical Report Bankable Feasibility Study Montagne d'Or Project French Guiana

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1 Summary

This report was prepared as a feasibility-level National Instrument 43-101 (NI 43-101) Technical Report (Technical Report) for Nord Gold SE (Nordgold) with Columbus Gold Corp. (Columbus) by SRK Consulting (U.S.), Inc. (SRK), on the Montagne d'Or Gold Deposit (Montagne d'Or or Project) located in French Guiana. Columbus is the Project owner and is currently exploring the deposit under an option agreement with Nordgold, the Project operator.

The Project will be a mining operation that will generally consist of a mine area (Mine), Process Plant, Tailings Storage Facility (TSF), Central Waste Rock Dump (CWRD), West Waste Rock Dump (WWRD) haul and access roads and ancillary facilities.

1.1 Property Description, Location and Ownership

Montagne d'Or is part of the larger Paul Isnard sector. The Project consists of eight mining concessions and two pending exploration permit applications covering a total area of 190 (square kilometres (km^2). The Project area and mining concessions are located in the northwestern portion of French Guiana, South America. The Project area extends from longitude 53° 53' 52'' W (Universal Transverse Mercator ((UTM) 178,475) to 54° 03' 09'' W (UTM 161,360), and latitude 4° 40' 59'' N (UTM 518,322) to 4° 51' 03'' N (UTM 536,922). The Project also includes historic artisanal mining operations, exploration roads, drill pads, a core logging/storage facility and a base camp. The Camp Citron base camp is located approximately 4 kilometres (km) northwest of the deposit. Columbus is the Project owner/operator and is currently exploring the deposit under an option agreement with Nordgold.

1.2 Geology and Mineralization

The Montagne d'Or deposit is a Paleoproterozoic age, high sulphidization, volcanogenic (VMS) gold deposit that has undergone remobilization and shear zone style deformation. The deposit is located within the northern greenstone belt of the Guiana Shield in French Guiana. Mineralization is hosted within the two billion year old, Paramaca Formation composed predominantly of meta-volcanic and meta-sedimentary units. These units have been deformed by ductile deformation resulting in tight to isoclinal folding and shearing as well as a pervasive foliation striking east-west and dipping steeply to the south. The current model of gold mineralization is a VMS type. Significant portions are thought to have been emplaced as replacement style mineralization. Subsequently, the mineralization has been deformed and partly remobilized within structural controls. Gold mineralization is associated with primary sulphide minerals as replacements within pyrite and chalcopyrite. At a macroscopic scale, the following five types of mineralization have been identified in mapping and drill core logging:

- Semi-massive sulphides (SMS) with >20% sulphides) with associated gold mineralization;
- Sulphides as disseminations and stringers with associated gold mineralization;
- Late-stage disseminated euhedral pyrite mineralization;
- Rhythmic mafic tuff with associated pyrrhotite mineralization; and
- Gold mineralization associated with quartz veins.

1.3 Status of Exploration, Development and Operations

The database supporting the resource estimation of this report is current to April 1, 2016. It contains information from 349 diamond core and reverse circulation drillholes and 87 channel samples. The drilling was completed in two main campaigns. A previous owner drilled 56 holes between 1996 and 1998. Columbus completed an additional 293 holes from 2011 to February, 2016. The channel samples were all collected from surface outcrops between 1995 and 1997. SRK has previously reviewed the 1995 through 1998 exploration data and found it to be of sufficient quality to support an industry standard, resource estimation. All drilling, sampling and analytical work conducted by Columbus has followed industry standard procedures and includes quality assurance/quality control (QA/QC) protocols.

1.4 Mineral Resource Estimate

Gold mineralization is controlled mainly by structural fabric and lithology. The mineralization is localized in planar zones which have recurrent distribution and highly variable grades. Anomalous gold grades typically occur in zones 3 to 10 metres (m) wide which are separated by barren or lower grade zones 10 to 30 m wide. As part of the most recent drilling campaign, most of the historic core was re-logged to create a unified system of lithologic descriptions. This has resulted in a detailed, 3-D geologic model created by using Leapfrog® Geo software. Lithologic control of mineralization is evident and SRK utilized four lithic types or groups which were estimated independently.

The gold (Au) capping level was chosen at 40 grams per tonne (g/t) resulted in 31 samples ranging from 40.1 to 163 g/t being reduced to 40 g/t prior to compositing. This capping results in a net loss of 3.4% of all gold in the database. Compositing was completed in 3 m downhole lengths with no breaks at lithologic contacts.

Columbus constructed generated wireframe solids with Leapfrog® software which enclosed anomalous gold mineralization at a 0.3 g/t Au threshold. The grade estimation was conducted in six domains. Three rock types/groups were used and each rock type/group was estimated independently both internal and external to the grade shell using only samples from the same domain. An Inverse Distance Weighting Squared (IDW^2) algorithm was used for the grade estimations.

Six techniques were used to evaluate the validity of the block model including; visual checks, overall model performance parameters, statistical comparison between composite and block grades, nearest neighbor comparisons, dilution sensitivity and swath plots.

The Mineral Resources reported by SRK for the Montagne d'Or deposit are classified as Measured, Indicated and Inferred Mineral Resources, based primarily on drillhole spacing since all other supporting data is of good quality. A wire frame solid was constructed around the area where the average drillhole spacing is approximately 35 m or less and these were used to assign the Measured Mineral Resource classification. This is a focused area of drilling completed in 2015 and 2016 located within the proposed Phase I pit. The measured wire frame solid is flanked by a second wireframe constructed around the areas where the average drillhole spacing is approximately 65 m or less and these were used to assign the Indicated Mineral Resource classification. All blocks outside of these wireframes were classified as Inferred Mineral Resources.

The Montagne d'Or Mineral Resource Statement is presented in Table 1-1. The resource is confined within a Whittle™ optimization pit shell and a Cut-off Grade (CoG) of 0.4 g/t Au applied. The pit shell and CoG assumes open-pit mining methods and is based on a mining cost of US$2/t, milling cost of US$15/t, administration cost of US$1/t, a gold price of US$1,300/oz., 95% gold recovery, gold refining cost of US$8/oz, and 5% net smelter return (NSR) royalty. A 45° pit shell slope was used for bedrock and a 35° pit shell slope was used for saprolite. The reported Mineral Resources include material from all estimation domains.

The effective date for the Mineral Resource estimate in this report is July 1, 2016 and was prepared by SRK. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Table 1-1: Montagne d'Or Mineral Resource Statement as of July 1, 2016, SRK Consulting (U.S.), Inc.

Classification	Au Cut-Off (g/t)	Tonnes (M)	Au (g/t)	Contained Au (Moz)
Measured	0.4	10.3	1.804	0.60
Indicated	0.4	74.8	1.350	3.25
M & I	0.4	85.1	1.405	3.85
Inferred	0.4	20.2	1.484	0.96

- All figures rounded to reflect the relative accuracy of the estimates.
- Metal assays were capped where appropriate.
- The Mineral Resources were estimated by Bart A. Stryhas PhD, CPG # 11034, a Qualified Person.
- Mineral Resources are reported based on a CoG of 0.4 g/t Au, and are reported inside a conceptual pit shell based on appropriate mining and processing costs and metal recoveries for oxide and sulphide material.
- CoGs are based on a mining cost of US$2/t, milling cost of US$15/t, administration cost of US$1/t, a gold price of US$1,300/oz., 95% gold recovery, gold refining cost of US$8/oz, and 5% NSR royalty.
- Silver was not included in the resource estimate. No gold equivalent grades are reported.

Source: SRK, 2016

1.5 Geotechnical

Two major geotechnical domains have been identified in the Project. A hard rock slope composed of strong foliated metamorphic rock and a near surface saprolite soil domain that controls the stability of the upper 30 to 40 m of the ground. The saprolite is a deeply and intensely weathered residual rock that behaves like a soil. It is weak, nearly saturated, and easily deformable.

SRK used Slide limit equilibrium program (RockScience, 2014) to assess the static slope stability for the pit slopes. The critical overall stability section is located on the south wall of the pit with a slope height of 308 m from the top of the pit slope to the pit bottom. Due to uncertainties in ground conditions, the stability analysis assumed a tension crack and piezometric level near the slope surface. Average strengths are used in the overall slope analysis. The critical stability surface has a minimum Factor of Safety (FoS) of 1.80 under these conditions and the potential failure surface would daylight at the toe of the pit slope. The critical surface runs predominantly through the felsic tuff and diabase dike units.

The saprolite slopes, being the weakest units, have the minimum FoS exceeding 1.3 on all sections analyzed. The saprolite slopes are the upper portion of the overall pit slope. Strengths for the saprolite have been developed from laboratory testing and back analysis of natural slope failures. The saprolite was analyzed for both average strengths, and a 25th percentile strength distribution value. The FoS assumes that the saprolite slopes are drained, given the design for drainage ditches at the saprock level along the pit walls. The stability of saprolite slopes is subject to the completion of a drainage design and placement of vegetative cover on all saprolite slopes following excavation. If undrained conditions exist or have not been covered with vegetation, the saprolite slopes are predicted fail by mechanisms of either erosion, flow, or creep.

Monitoring of slopes will be required due to the uncertainties in conditions. A slope monitoring program should be implemented at the pre-mining stage of the Project. The program should be used to identify any incipient failures (including natural slope movements up dip of the ultimate pit walls) and determine the course of action, which could include unloading or buttressing of slopes.

Several geotechnical risks have been identified for the Project that have been incorporated into the Project risk register. These risks include: existing natural landslide hazards above the pit slope to the south of the pit, potential for slope creep under sustained wet conditions, flow and erosion of the saprolite if slope drainage measures are not effective, potential for high groundwater levels in the rock slopes; and rockfall and multi-bench failures in the pit slopes. Mitigation of these risks have been addressed as a part of the slope BFS design and stability criteria in the study, and the recommended slope monitoring program. As mining commences additional risk reduction may be accomplished by conducting geologic and geotechnical mapping and analysis.

1.5.1 Waste Rock Stability Analysis

SRK used Slide limit equilibrium program (RockScience, 2014) to assess the slope stability for the Waste Rock Dump (WRD) slopes. The predicted minimum FoS is 1.40 for the WRD slope design. The critical surface is located on the 20-m high berm at the base of the WRD (at 36° slope angle, or 1.4 horizontal to 1 vertical (1.4H:1V)). The critical surface is predicted to pass through the saprolite foundation extending to the crest of the dump slope. The overall predicted slope FoS is 1.70 for the 100 m high dump slope with an overall slope angle of approximately 24°(2.2H:1V). The FOS values for both critical sections exceeded the minimum required FOS of 1.3.

1.6 Mineral Reserve Estimate

Life-of-Mine (LoM) plans and resulting Mineral Reserves are determined based on a gold price of US$1,200/oz Au. Reserves stated in Table 1-2 are dated effective as of September 1, 2016 with a Euro:USD exchange rate (EURUSD) of US$1.10:€1.00.

The ore material is converted from Mineral Resource to Mineral Reserve based primarily on positive cash flow pit optimization results, pit design and geological classification of Measured and Indicated resources. The in situ value is derived from the estimated grade and certain modifying factors.

The Qualified Person has not identified any risk including legal, political or environmental, that would materially affect potential development of the Mineral Reserves, as of September 1, 2016.

Table 1-2: Montagne d'Or Mineral Reserve Estimate as of September 1, 2016, SRK Consulting (U.S.), Inc.

Class	Tonnes M	Au g/t	Contained Au Moz
Proven	8.25	1.99	0.53
Probable	45.87	1.50	2.22
Proven and Probable	**54.11**	**1.58**	**2.75**

- Mineral Reserves are reported at varied cut-offs dependent on lithological rock types, economics and estimated metallurgical recovery. Felsic Tuffs have CoG of 0.617 g/t Au, Granodiorites have a CoG of 0.622 g/t Au, Mafics have a CoG of 0.665 g/t Au, Saprolite and Saprock have a CoG of 0.552 g/t Au.
- Associated metallurgical recoveries have been estimated as 93.8% for Felsic Tuffs, 95.2% for Granodiorites, 91.3% for Mafics and 96.4% Saprolite/Saprock
- Full mining recovery assumed.
- Reserves have no additional dilution added to that that inherent in the selective mining unit (SMU) of 5 m x 5 m x 5 m diluted mine block model.
- Reserves are based on a US$1,200/oz Au gold price.
- Reserves are converted from resources through the process of pit optimization, pit design, production schedule and supported by a positive cash flow model.
- The ore reserves were estimated by Bret C Swanson, BE (Min) MMSAQP #04418QP, a Qualified Person.
- Silver was not included in the reserve estimate. No gold equivalent grades are reported.
- The reserves are valid as of September 1, 2016.

1.7 Mining Methods

1.7.1 Mine Planning

The Project is located on the side of a moderately sized hill, surrounded by dense tropical rainforest in a remote location that has been disturbed by historic illegal mining. The Montagne d'Or mine will be an open pit mine that uses gravity/cyanidation as the primary method of extracting gold from the Mineral Resource. Through the process of pit optimization, pit design, production scheduling, and capital and operating cost estimation, the conversion of Mineral Resources to Mineral Reserves resulted in a diluted reserve of 2.75 Moz Au at 1.58 g/t Au defined in situ before metallurgical recoveries.

The Bankable Feasibility Study (BFS) open pit is approximately 2.5 km long by 500 m wide, and of varying depth from surface[1], with a stripping ratio of 4.5 to 1 (waste to ore). (Note 1: The open pit is located on the side of a hill. The average pit north wall is approximately 125 m deep from original ground surface, and the average pit south wall is approximately 225 m in height. The pit centroid depth from original ground surface is 185 m). Figure 1-1 illustrates the planned pit, WRD and TSF locations for the Project.

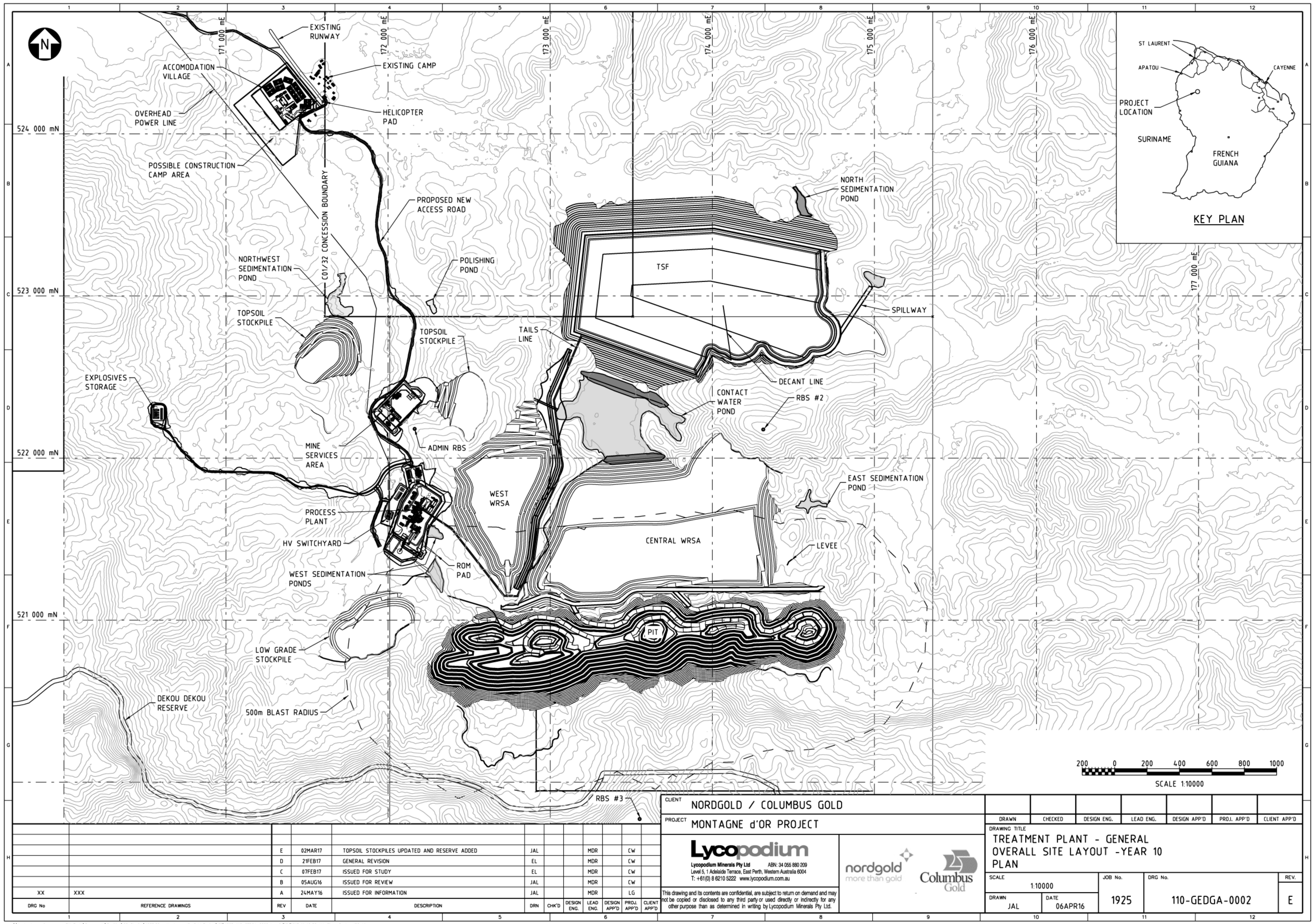


Figure 1-1: Montagne d'Or Site Layout

The mine production schedule is based on feeding the processing facility operating at a rate of 12,500 tonnes per day (t/d) or approximately 4.6 million tonnes per year (Mt/y) of mill feed. The mill feed was separated into three CoG's that represent the internal CoG for gold prices of US$400/oz, US$800/oz and US$1,200/oz, and includes multiple recoveries ranging from 90.3% to 96.4% dependent on rock types, for the purpose of the CoG calculations.

The targeted mining rate is approximately 80 thousand tonnes per day (kt/d) (waste and ore), which provides a higher mill feed rate than the mill can process, requiring mill feed stockpiles to be used to store the excess. The use of stockpiles ensures that the highest grade mill feed is sent to the crusher before lower grade is processed. This creates a variable cut-off that defers marginal mill feed that will be processed at the end of the mine life, thus optimizing the Project net present value (NPV) and cash flow. The maximum stockpile size is approximately 8 million tonnes (Mt) of material. Mining rates have been adjusted by up to 30% to account for the wet and dry seasons that will be encountered during operations.

Dilution has been incorporated into the mine block model for the BFS. As there is no operational history, dilution was calculated by determining the partial quantity of gold units within and outside the grade shell used for resource interpolation. The diluted grade for the model is referenced to a 5 m x 5 m x 5 m block dimension that represents the Selective Mining Unit (SMU) assumed for the BFS. This is supported by the planned drilling pattern of 5.1 x 5.1 m representing grade control definition.

1.7.2 Mining Operations

SRK assumed that open pit mining methods will use front-end loaders (FELs) and hydraulic excavators to load haul trucks for waste and ore haulage. Mining activities will include site clearing, removal of growth medium (topsoil), free-digging, drilling, blasting, loading, hauling and mining support activities. Material within the pit will be generally blasted on a 5 m high bench. Most of the saprolite material (approximately 18% of the total material to be mined) can be loaded directly with hydraulic excavators without the need for blasting. Most ore will be sent directly to the primary crusher. The stripped waste material will be placed in dumps to the north of the pit, and lower-grade ore placed in a stockpile, near the primary crusher location.

Because of the large amount of rainfall, hilly terrain, and amount of saprolite, SRK developed a mixed mining fleet. The first fleet was comprised of 6.7 cubic metre (m^3) capacity excavators that loaded 40 t articulated dump trucks (ADT). This first fleet will be used for pioneering excavation, most of the saprolite mining and can also assist with selective ore mining. As the majority of saprolite is removed and drainage improved, the second larger mining fleet of 12.0 m^3 capacity excavators and 91 t capacity rear dump trucks will perform the majority of the bulk production.

The mine equipment requirements and costing were based on the purchase of new equipment. It was planned that all mine mobile equipment would be diesel-powered, to avoid the requirement to provide electrical power into the pit working areas. The mine operations schedule is proposed to include two 12-hour shifts per day, seven days per week for 355 days per year This includes an annual allowance of 10 days downtime for weather delays for most of the mine operations, and 15 days downtime for weather delays for the drilling operations.

An explosives provider for the mine will have explosives storage facilities at the mine site, located to the west of the Mine Services Area (MSA). The explosives provider for the mine will also be the blasting contractor for the mine. Commencing at the same time as the mill production (start of Year

1), the blasting contractor will start production of bulk emulsion using an emulsion plant located within the explosives storage facilities compound, which will be capable of sufficient bulk emulsion production over the life of the planned mining operations.

Table 1-3 shows the major mining equipment requirements for selected years of the mine plan. Years -2 and -1 are the pre-production mining operations. The Project mining schedule has set year 2020 (Yr -2) and 2021 (Yr -1) as the pre-production mining years, with production mining starting in 2022 (Yr 1).

Table 1-3: Planned Major Mining Equipment Fleet for Selected Years

Equipment Units	Make	Model	Size	Yr -2	Yr -1	Yr 1	Yr 3	Yr 5	Yr 7	Yr 9	Yr 11	Yr 12
Drilling												
Blasthole drill	Atlas Copco	SROC D65	152 mm	1	2	3	4	4	4	4	1	-
Loading												
Front end loader	Komatsu	WA600-8	6.4 m^3	1	1	1	1	1	1	1	1	1
Front end loader	Komatsu	WA800-3EO	12.3 m^3	-	-	1	1	1	1	1	1	-
Hydraulic excavator	Komatsu	PC1250LC-8	6.7 m^3	2	2	2	2	2	2	1	-	-
Hydraulic excavator	Komatsu	PC2000-8	12.0 m^3	-	-	3	3	3	3	3	2	2
Hauling												
Haul truck - ADT	Komatsu	HM400-5	40 t	8	8	9	9	3	5	4	-	-
Haul truck – Rear dump	Komatsu	785-7	91 t	-	-	13	15	17	17	17	4	4
Other Mine Equipment												
Crush/Screen Plant	Manufacturer	Jaw/Cone/Screen	335 kW	1	1	1	1	1	1	1	-	-
Track dozer	Caterpillar	D10T	447 kW	4	4	4	4	4	4	4	1	1
Wheel dozer	Caterpillar	834H	372 kW	-	-	1	2	2	2	2	1	1
Motor grader	Komatsu	GD675-6	165 kW	3	3	4	4	4	4	4	2	2
Backhoe loader	Caterpillar	450E	102 kW	1	1	1	1	1	1	1	-	-
Water truck	Scania	P410CB 8X4	30,000L	2	2	3	3	3	3	3	3	1
Excavator	Komatsu	PC800LC-8	363 kW	2	2	2	2	2	2	2	1	1
Compactor	Caterpillar	CS/CP 533E	97 kW	2	2	2	2	2	2	2	1	1

Source: SRK, 2017

Pit waste quantities of saprolite and rock will be used in construction of the TSF embankments in particular years. The waste haulage costs for these were included in the mining costs. A separate construction equipment fleet will be used for Project construction work.

Dewatering will be required for the open pit. Precipitation inflow directly into the pit and pit groundwater inflow will be collected at the bottom of each pit phase in a series of sumps, and pumped to the to the pit rim and from there channelled in accordance with contact water flows. Most precipitation falling outside of the perimeter at the top of the pit will be diverted around the pit into various drainages.

1.8 Mineral Processing and Metallurgical Testing

The metallurgical program for the Montagne d'Or BFS was based on earlier metallurgical studies that were conducted as part of a Preliminary Economic Assessment (PEA) of the Project during 2014 and 2015 by Bureau Veritas Commodities Ltd – Inspectorate Metallurgical Division (BV) and documented in their report, "Metallurgical Testing to Recover Gold and Silver on Samples From the Montagne d'Or Project, French Guiana, April 6, 2015." The PEA metallurgical program evaluated three process options, including whole-ore cyanidation, a combination of gravity concentration followed by cyanidation of gravity tailing, and gravity concentration followed by gold flotation from the gravity tailing and cyanidation of the flotation concentrate. Based on the results of the PEA, the BFS metallurgical program focused on the development of a process flowsheet that included gravity concentration followed by cyanidation of the gravity tailings and intensive cyanide leaching of the gravity concentrate. This program was conducted by several different commercial laboratories including: BV, Pocock Industrial, ALS Metallurgy – North America (ALS), SGS Canada, and FLSmidth.

The metallurgical program was conducted on three master composites, 15 variability composites representing different ore lithologies and grade ranges, and seven variability composites representing seven mining phases that were identified at the start of the program.

The following significant factors are identified based on the metallurgical studies conducted for the BFS:

- The BFS metallurgical program focused on the development of a process flowsheet that included gravity concentration followed by cyanidation of the gravity tailings and intensive cyanide leaching of the gravity concentrate;
- Montagne d'Or ore can be readily processed to recover the contained gold and silver values using unit operations considered standard to the industry;
- SRK has estimated overall adjusted gold and silver recoveries based on the contribution from each ore lithology during each phase of mining. During the first six mining phases gold recovery is estimated at 94% to 95% and silver recovery is estimated at about 54% to 56%. These recovery projections include a 2% deduction from reported laboratory test results to account for inherent plant inefficiencies; and
- Detoxification of the cyanide leach residues was accomplished with the industry-standard Sulfur Dioxide (SO_2)/Air process. It was demonstrated that cyanide in the leach residue could readily be detoxified to less than 1 part per million (ppm) Weak Acid Dissociable cyanide (CN_{wad}). SO_2 consumption in the range of about 5 to 6 g Sulfur Dioxide per gram (SO_2/g) CN_{wad} were reported, which is typical of industry practice.

1.9 Recovery Methods

The process plant design, derived from the interpretation of the test work results, reflects a robust metallurgical flowsheet designed for optimum recovery with minimum operating costs and utilising unit operations that are well proven in industry. The key criteria for equipment selection are suitability for duty, reliability and ease of maintenance. The plant layout provides ease of access to all equipment for operating and maintenance requirements whilst maintaining a compact footprint that will minimize construction costs.

The key Project and ore specific criteria for the plant design are:

- 4.6 Mt/y (12,330 t/d) throughput based on the design ore blend of 89% felsic tuff, 7% granodiorite and 4% mafic;
- Mechanical availability of 91.3% supported by crushed ore storage and standby equipment in critical areas; and
- Sufficient instrumentation and automation to achieve design production rates, to enable stable process operations and to facilitate safe operation.

The Montagne d'Or plant has been designed to treat the range of ore types and blends that will be mined over the life of the Project.

The treatment plant design incorporates the following unit process operations:

- Primary jaw crushing.
- A crushed ore surge bin with bin overflow conveyed to a dead stockpile.
- A single stage Semi-autogenous Grinding (SAG) mill in closed circuit with a pebble crusher and hydrocyclones to produce an 80% passing 75 micron (P_{80} 75 µm) grind size.
- Gravity concentration with intensive cyanidation and electrowinning of recovered gold.
- Pre-leach thickening.
- Leach/Carbon in Leach (CIL) circuit incorporating a leach tank and six CIL tanks.
- A 10 tonne split Anglo American Research Laboratories (AARL) elution circuit.
- Tails wash thickener.
- SO_2/Air cyanide destruction circuit to reduce the tailings CN_{WAD} concentration to below 10 ppm.
- Tailings pumping to the TSF.
- Supporting air and water services and reagent and consumables handling.

1.10 Project Infrastructure

Existing infrastructure at site is minimal to non-existent. The Project is accessible via a 120 km seasonal forest road from the town of Saint Laurent du Maroni (SLM), where the port of St. Laurent is located, or by helicopter/light aircraft to the Project's base camp at Camp Citron.

The current condition of the public section of the road between SLM and Apatou Crossing road is fair to poor and will need repair and maintenance during the Project construction and on-going operation phase.

Infrastructure to be provided to support construction and operation includes:

- Rehabilitation of the existing 54 km of road between the Project site and Apatou Crossing;
- Site roads and earthworks pads for the construction of site infrastructure;
- Stormwater management and sediment control structures;
- Contact Water Pond (CWP) to store all potentially contaminated site water for use in the process plant and/or for treatment prior to discharge;
- Construction of a 120 km 90 kiloVolt (kV) overhead power line to connect the Project to the national power grid at SLM;
- Expansion of the existing Camp Citron to provide pioneer accommodation for early Project construction activities;
- Construction of a 482 room permanent camp to support construction and operations including potable water and sewage treatment plant, waste disposal facilities and temporary power;
- Site communications including an external voice/data link and internal local area network (LAN) and radio network as well as site mobile phone coverage;
- Administration infrastructure such as offices, clinic, emergency response, warehouses, site laboratory etc.;
- Mine support services including offices, ablutions, workshops, fuel depot, explosives facility etc.;

- Plant support services including security and access control, offices, ablutions, control room etc.;
- A lined TSF capable of being progressively expanded to contain the LoM tailings from the process plant;
- Water Treatment Plants (WTPs) to raise the quality of surplus site contact water and TSF decant water to a level where it is suitable for discharge into the local watercourses;
- Temporary topsoil dumps for use for site rehabilitation during and after the mine life; and
- WRDs for the permanent management of mine waste.

1.11 Tailings Storage Facility

The principal objectives of the TSF design are to protect the regional groundwater and surface waters during operations and closure, provide secure storage for 56 Mt of tailings, provide a development plan that utilizes four construction phases in order to minimize initial capital expenditures, and meet closure objectives.

The TSF will consist of two embankments separated by north-south trending ridges. The embankments will be raised in phases using the downstream construction method. While this method requires the largest embankment fill volume, it provides the most stable embankment configuration in terms of static and seismic loading because the embankment fill is founded on competent foundation soils or bedrock.

The embankments will be constructed with 2.5H:1V upstream and downstream slopes, with a 17 m crest. In order to meet the minimum stability criteria, up to 5 m of the foundation soils will need to be removed beneath each of the embankments, including part of the South Embankment abutment. The tailings embankment will be constructed over four phases to minimize initial capital expenditures and defer the additional expenditures to the extent possible.

An assessment of the geochemistry of the tailings indicates they will have a strong acid generating potential (AGP). Based on this determination and residual presence of reagents in the tailings effluent, the TSF will need to be lined. The liner will consist of a single 2.0 mm Linear Low Density Polyethylene (LLDPE) geomembrane over a prepared subgrade surface within the entire TSF impoundment. While High Density Polyethylene (HDPE) and LLDPE geomembranes are the most common lining materials, a LLDPE geomembrane was selected due to its higher puncture resistance and greater elongation properties.

An underdrain system will be installed to protect groundwater and minimize any uplift pressures on the geomembrane liner system. The underdrain will be comprised of a free draining granular material in order to collect groundwater associated with of any springs or seeps within the TSF footprint. Water captured by the underdrain will flow via gravity to a sump north of the TSF footprint.

Consideration was given to the installation of an "internal" leak detection system. However, given the capital costs and schedule impacts of installing a double liner system with a granular drainage layer, a leak detection system was not included in the TSF design. Despite the absence of a dedicated leak detection system, it is assumed that the proposed underdrain system will intercept the leaked supernatant and direct it to the underdrain sump. Water reporting to the underdrain sump will, depending on its quality, either be discharged to the environment or pumped back into the TSF.

Slurried tailings will be pumped from the mill to the impoundment via the tailings delivery pipeline, and then to planned deposition locations via the tailings deposition pipeline. Deposition will occur sub-aerially and will initially be performed mainly from embankment deposition points to push tailings and entrained water away from the embankment and simultaneously establish deposition cycles that optimize the creation and maintenance of a well-drained beach with a positive gradient to the southwest (i.e., away from the embankments). The water balance estimates that a makeup demand of approximately 140 L/sec is required. The majority of this demand is satisfied by the TSF, i.e. 180 to 120 L/sec is typically provided by the TSF. The remainder of the demand will be provided by water stored in the CWP or precipitation captured in the plant and mill bund area and made available for raw water makeup.

To support the surface water management design, SRK included TSF diversion channels on the south side of the TSF impoundment to intercept and divert surface water from the hillslope above the TSF and a Downchute Channel to discharge surface water from the TSF Perimeter Diversion Channel to the North Sedimentation Pond for Phase 1 through Phase 3. A Closure Spillway will be constructed in Phase 4, once the TSF Embankment has been constructed to its ultimate elevation.

1.12 Site Water Management

1.12.1 Hydrogeology

During operation, significant volumes of surface run-off and shallow groundwater from the drainages where saprock is exposed will be captured in a diversion ditch along the top of the pit, to minimize the volume of water reaching the exposed rock in the open pit. The diversion water will be routed to sediment control ponds and then to undisturbed creeks. However, groundwater in bedrock and in faults and joints within the bedrock will report to low points in the open pit and require pumping to a CWP. Because the intact bedrock is of low hydraulic conductivity, the relative contribution of groundwater reaching the open pit will be less than that of surface water run-off reporting to the pit. Consequently, it is unlikely that active dewatering of the bedrock or saprock with dewatering wells around the pit perimeter will reduce costs or significantly improve long term mining conditions in the open pit. Groundwater reporting to the open pit will mix with run-off and direct precipitation and collect in sumps in the low areas of the pit; this water will be pumped out of the pit with a set of sump pumps, and directed to managed ponds and creeks as described in Section 1.12.2.

1.12.2 Site Water Management

The Project is located in an area of high rainfall, therefore it is anticipated that the system will consistently experience high intensity short duration stormwater. Additionally, low intensity contact water inflows will result in a steady inflow of water to the mine facilities.

The mine water management plan addresses stormwater and mitigates much of contact water inflows by diverting as much clean, non-contact water from adjacent hillsides around the Project facilities. Where mine sequencing and the topography allows, diversion ditches have been designed upgradient of the pit, WRDs, stockpiles, and TSF to minimize the amount of water that runs on to the facilities. Non-contact stormwater will be monitored for sediment loading and discharged when meeting applicable water quality standards.

Water that cannot be diverted will come into contact with active mining facilities and becomes contact water, which is managed separately from non-contact water to avoid release to the environment. Additionally, seepage from waste rock and ore, run-off from active WRDs and stockpiles, run-off from pit walls, and seepage from groundwater to the pit sumps will also contribute to contact water. All contact water will be isolated and routed to the CWP. From the CWP, contact water may be consumed as raw water in the mining process but modeling has indicated that the amount of contact water produced is far in excess of the demand for raw water. Excess contact water will be discharged to the environment which may require treatment to meet applicable standards. The CWP has been sized to prevent uncontrolled releases to the environment as a result of high rainfall, and to store sufficient water to supply the mining process with makeup through periods of extended drought.

Water that has come into contact with process activities, such as the TSF, will be contained to prevent release to the environment. The capture of precipitation falling on the TSF will produce process water that, under most conditions, will exceed the amount of water consumed in the milling and tailings deposition process. Water balance modeling indicates that after the early stages of the mine life, there will be a net accumulation of process water within the circuit, requiring that excess process water be removed from the circuit on a regular basis, utilizing a treat and discharge approach that allows water to be discharged to the environment.

1.12.3 Geochemistry

Based on geochemical laboratory analyses of metallurgical test products, tailings pore water will be pH neutral when first discharged, but tailings solids will be net acid generating due to the presence of 1.2% sulphide (dominantly pyrite with subordinate chalcopyrite and pyrrhotite) with negligible associated carbonate. Minimizing the production of acidic pore water from the pyrite and other sulphides in the tailings solids can be achieved by maintaining complete submergence of the tailings solids under a water cover. Areas of intermittent wetting and drying, such as beaches and embankments, could be conducive to production of acid rock drainage and metal leaching (ARDML).

Geochemical characterization data indicate that approximately 41% of the waste rock will be comprised of the rock types felsic tuff and lapilli tuff, which are categorized as net potentially acid generating (PAG). Due to the dominance of non-PAG waste rock excavated in pre-production years, WRD drainage in years -2 and -1 is predicted to be circum-neutral. However, as the volume of excavated PAG rock increases disproportionately compared to non-PAG rock, waste rock drainage pH is predicted to decline and persist in the range of 3 - 3.5 until the end of mining. A closure strategy of cover emplacement concurrent with waste rock deposition, in conjunction with a material handling and segregation plan, could significantly attenuate the production of acid rock drainage from waste rock.

Geochemical predictive modeling calculates that the pit lake will maintain a slightly alkaline pH (~8.1) through all stages of infilling and into closure with all metal concentrations below regulatory limits. This is primarily because of the prediction from the groundwater flow model that the pit lake will fill rapidly (73 months in base case) with dilute groundwater and surface water, which will minimize sulphide oxidation and weathering of pit wall rock minerals.

1.13 Environmental Studies and Permitting

1.13.1 Known Environmental Liabilities

Environmental liabilities resulting from previous and ongoing exploration activities are fairly limited due to the high precipitation and rapid natural revegetation that occurs in the rainforest. Illegal artisanal placer mining occurs over much of the Project area has disturbed considerable land area, and continues to impact local surface water resources through sediment release and water contamination.

1.13.2 Environmental Studies

A number of technical environmental studies have been conducted as part of Project development, many of which were prepared as part of the *Montagne d’Or Gold Project Environmental Scoping Study* (WSP, 2015a). These studies are intended to provide direction for the environmental assessment process, and guide the environmental authorities with the information required to determine the range of information and degree of detail needed in the formal impact assessment. Studies have been completed in the following areas, with key points as follows:

- Biological Reserves and Resources. The mining Project is located between the two sections of the Integral Biological Reserve (RBI) of Lucifer Dékou-Dékou, in a space designated as Managed Biological Reserve (RBD). The Lucifer and Dékou-Dékou massifs are home to two floral assemblages rare in French Guiana: the sub-montaneous forest on lateritic bauxite hardpan, and the forest on 400 to 500 m slopes. They shelter some fifty floral heritage species and three nationally-protected species. This heritage value led to the creation in 2012 of the Lucifer and Dékou-Dékou RBI, the first such reserve in French Guiana and the largest in French jurisdiction. Within the RBI, any direct human intervention that could modify the functioning of the ecosystem is prohibited. The only authorized sylvicultural measures are those eliminating exotic or invasive species and the securing of trails and roads bordering or crossing the reserve.

  While the Project itself is located in portions of a RBD, mining activity is permitted under certain conditions. This exception was established to take into account historic exploration and exploitation of gold resources in the area, as well as the presence of potentially significant mineral deposits at the foot of the Dékou-Dékou massif.

  On the basis of the principle ‘avoid-reduce-offset’, optimization measures of the Project have been developed in order to avoid impacts on biodiversity, including the elimination of the WRD to the northeast of the pit in order to preserve the wildlife migration corridor. Measures to reduce the impact will be also prescribed in the impact assessment study. In addition, a compensation program tailored to the scale of the Project and the challenges of biodiversity is underway with the local partners in order to compensate for residual impacts on biodiversity.
- Threatened, Endangered, and Special Status Species. A total of 110 nationally protected species were recorded on the site of which 100 bird species including three species with protected habitat, seven mammals and three plants. The site also hosts five plant species new to French Guiana and seven other plants of interest (rare or endemic), as well as two fish species rare and endemic to French Guiana, present on the mountain creeks.

- Air Quality. The overall air quality is good, given the lack of human activity in the area and the dense forest cover. As a result, the sensitivity regarding air quality will likely be high, especially since the RBI including the Dékou-Dékou massif, to the south of the Project, and the Lucifer massif, to the north of the Project, must be preserved.,
- Cultural and Archeological Resources. Pedestrian survey campaign has ended with the discovery of 47 proven sites attributed to the pre-Columbian period, and fifteen 'crowned mountains' including 10 sites that are spread over an area of about 40 km² around the future Project. To the extent practicable, these locations are avoided by the mine plan.
- Land Use. Most land (including the access road between SLM and Citron Camp) consists of wet lowlands forest,
- Hydrogeology (Groundwater). Hydrogeological modeling in the area of the open pit predicts that, at closure, the pit will fill with water and start overtopping in approximately 6 years. This rapid influx of water will have the effect of introducing a large volume of relatively clean water in a short time period, which results in reasonably good pit lake water chemistry sustained into the future. The risks of creating a low-quality pit lake post closure are considered minimal;
- Hydrology (Surface Water) and Water Quality. The site is located in region of high rainfall. As such, stormwater management and diversion will be critical to Project success, and excess waters will necessarily require treatment and discharge in order to maintain an appropriate site-wide water balance, and
- Waste Rock Geochemistry. The results of the static testing program indicate that approximately 55% of the waste rock is classified as PAG, approximately 30% is classified as non-PAG or non-acid generating (NAG), with the remaining fraction (~15%) classified as uncertain. Kinetic Net Acid Generation (KNAG) test data, however, indicate that only the Felsic Tuff and the Lapilli Tuff are PAG. All other rock types are net non-PAG due to encapsulation of sulphides by quartz and other silicate phases which renders the sulphide minerals unreactive. The consequence of these tests is a reduction of the fraction of PAG waste rock from 55% to 41%. This is a significant finding that indicates that the mass of acid generating waste rock is considerably less than indicated by the Acid Base Accounting (ABA) results, which has important implications for waste rock management plans. In light of these results, the potential for leaching metals remains a concern at this stage, and will need to be considered during detailed design and construction of the mine. The waste rock, low grade ore, and tailings management will be subject to the guidance of the BAT-Management of Tailings and Waste-rock in Mining Activities (MTWR, 2009) and will likely follow recommendations from the draft Management of Waste from Extractive Industries (draft MWEI, 2016) in order to comply with French PAG waste storage regulations.
- French guidelines classify waste rock with a neutralization potential ratio less than three (NPR<3) as potentially hazardous material. As approximately 50% of the waste rock for the Project has a NPR of three or less (under the current classification program), this may require part or all of the WRD to have a barrier layer with a permeability of 1×10-9 metres per second installed to minimize seepage to groundwater, and could potentially increase the Project capital. SRK developed the current base case capital cost estimate without a WRD barrier system. SRK assumed that: (1) additional waste rock geochemical characterization and classification would reduce the quantity of waste rock that could be classified as potentially acid generating (PAG) and, as such, reduce the need and extent of such a barrier

system; (2) additional engineering and field investigation would be required to support a practical demonstration (or equivalency thereto) of containment of WRD seepage from the PAG materials. This demonstration could be made in several ways, including: demonstration that the existing in-situ saprolitic soils have an equivalent containment to protect groundwater; that amended saprolite (i.e., bentonite geosynthetics, treated soils, sand-bentonite-polymer, etc.) and/or a geomembrane could be used for an equivalent barrier system; (3) that the attenuation capacity of the underlying saprolite soils could meet the regulatory requirements; and/or, (4) that permitting discussions of selective handling, placement, and encapsulation of potentially reactive waste (common industry practice) could meet regulator needs and eliminate the requirement for a soil/geomembrane barrier system. This regulatory requirement may also extend to the low-grade ore stockpile. As such, there is a risk that a waste rock and low-grade ore barrier systems may need to be added to the Project capital cost estimate.

- Tailings Geochemistry. The detoxified tailings solids are likely to be net acid generating. The supernatant will initially be alkaline when first discharged to the TSF, and should aid in buffering the overall system.

The detoxified tailings solids are likely to be net acid generating. The supernatant will initially be alkaline when first discharged to the TSF, and should aid in buffering the overall system.

1.13.3 Stakeholder Engagement and Principal Issues

Initial stakeholder consultation was performed by WSP between September 15 – 19, 2014, in Cayenne and Saint-Laurent-du-Maroni. These meetings highlighted potential environmental and social issues associated with the Project and for which a public, professional or legal concern may arise. WSP (2014) summarized the main issues and concerns expressed by stakeholders during a first series of consultations, which include, but are not limited to:

- Integrity of the Lucifer Dékou-Dékou/RBI;
- Protection of flora and fauna and quality of biological inventories;
- Proactive and transparent communication with stakeholders;
- Local and regional jobs and economic spinoffs;
- Training of qualified local workforce;
- Contribution to the fight against illegal gold mining;
- Sound environmental management;
- Prevention of pollution and industrial risks, including those related to the eventual use of cyanide;
- Protection of watersheds; and
- Workplace health and safety.

1.13.4 Project Permitting

WSP (2015) provides a preliminary identification of the regulatory elements to which the Project is subject, based on information currently available. Most of the Paul Isnard (Nordgold) concession areas, including the Montagne d'Or gold deposit, lie within Zone 2 as defined by the *Schéma Départemental d'Orientation Minière* (SDOM) legislation adopted in 2012. Some of the conditions for mining in Zone 2 include:

- Demonstration of a viable mineral deposit;
- Completion of an Environmental Impact Study and Reclamation Plan; and
- Possible additional reclamation or environmental investigations, as may be required for the public interest, on or off site.

In addition to the land restrictions presented by the SDOM, the Project is located adjacent to a nature reserve, the Integral Biological Réserve Lucifer Dékou-Dékou, managed by the *Office National des Forêts* (ONF) [French National Forestry Board]. Its Management Plan from the ONF is yet to be developed, so there is little guidance or decisions regarding the use of land and allowable activities within the reserve. The boundaries of this reserve overlap four of the eight Project mineral concessions; however, only one of these concessions is important to the Project. Since these concessions already exist, and there has been continued exploration and mining activity in the area for over 100 years, the ONF has agreed to create several zones within the reserve boundaries where mining is permitted. The Montagne d'Or deposit itself is within a zone where open pit mining is permitted and the outer limit of the pit design is located at least 440 m from the reserve boundary.

French Guiana's mining regime is governed by the legislative and regulatory regime applicable to the French mainland with the exception of certain legal and regulatory provisions which are specific to it in order to take into account particular characteristics and constraints of this overseas territory. The Mining Code requires that two conditions be met in order to be able to explore or exploit a Mineral Resource: holding a mining title (provided at a national level); and obtaining work authorizations (at a territorial level).

The general provisions of the Mining Code provide for two main types of mining titles: the exclusive exploration permit ('*permis exclusif de recherche*' or PER) for the exploration phase, and the concession (Concession) for the exploitation phase. In addition, small-scale mining, including most lawful alluvial operations, are carried out through exploitation authorizations ('*autorisation d'exploitation*' or AEX) granted for areas no larger than one km^2. There are no current AEX operations within the Project area.

The Project is linked to the exclusive right through the Concession No 215 - C02/46, held by SOTRAPMAG, a subsidiary of Columbus, on which the Montagne d'Or deposit is located. This concession was granted on May 21, 1946 (J.O. of June 1, 1946) to S.E.E.M.I., and subsequently transferred to SOTRAPMAG by the Decree of December 27, 1995 (J.O. of December 29, 1995). This mining concession, combined with the appropriate work permits, allows large-scale mine operations and is valid until December 31, 2018. An application was submitted to the Minister in charge of mines on December 2016 to request a first 25-year extension. Concession renewal is subject to conditions, not the least of which is proving economic viability. Two exploration permits (PER) (identified as Cigaline and Bernard), at the western and eastern limits of the Concession C02/46, were obtained on July 13, 2016. These PER cover 189.5 km^2 and will be used to explore the eastern and western extensions of the Montagne d'Or deposit.

On March 13, 2014, Columbus and Nordgold signed the definitive option agreement pursuant to which Nordgold has the right to earn a 50.01% interest in the Project and the pending PER applications (54.3 km^2) within a three-year option period terminating in March 2017.

The French Environment Code has specific regulations for facilities (including mining operations) which may present dangers or inconveniences for neighbours, health, safety, public hygiene or the environment. These *Facilities Classified for Environmental Protection*, or ICPE, are subject to authorization, registration. In addition, the Project will be subject to European Directives on industrial emissions, which includes, in many cases, the use of Best Available Techniques (BAT) for the subject activities.

It is currently envisioned that the Montagne d'Or permitting process will require at least two years to complete for the mine, plant, and explosives emulsion plant. Each major permit application must include an Environmental Assessment (EA) which includes Avoid-Reduce-Compensate measures, and a specific focus on endangered species; a Hazard Study (HS) evaluating major risk scenarios for the Project define preventive and protective measures; as well as relevant technical studies supporting the findings of the EA and HS.

1.13.5 Reclamation and Closure

Upon final closure, the operator is required to provide an assessment of the final soil and groundwater conditions in comparison to the previously developed IED baseline studies report developed by Geoplus Environment (2017). The operator is required to restore the site to a state that is, at a minimum, similar to that described in the baseline report and suitable for the selected future land use.

The objective of reclamation activities is to provide long-term stability, waste containment (to avoid both migration of pollutants and waste and minimize the risk of oxidation, leachate generation, and release of heavy metals), and erosion prevention to reduce impact on the environment per the French Environment Code, Directive 2006/21/EC on the management of waste from extractive industries, and IED Directive concerning integrated pollution prevention and control. In order to achieve 'feasibility' at this early stage of the Project, reclamation and closure of the earthworks facilities will be in accordance with the 'Order of 15 February 2016 relating to non-hazardous waste storage facilities' and BAT Reference Document for the Management of Waste from the Extractive Industries (draft document, June 2016). Following the development of the Environmental and Social Impact Assessment (ESIA), and associated environmental management plans, Nordgold may have an opportunity to modify these closure approaches during detailed design when more information has been developed, and equivalent levels of environmental protection can be effectively demonstrated.

1.14 Capital and Operating Costs

Based on an EURUSD of US$1.05:€1.00, total capital costs totaling US$827 million including contingency and final closure/reclamation costs are summarized in Table 1-4. Approximately 9.5% overall contingency has been applied to capital items, which is appropriate for a BFS level of analysis in SRKs opinion. The initial capital required to construct a 4.6 Mt/y project that will produce approximately 237 thousand ounces per year (koz/y) during the first 10 years of the operation is estimated to be US$535.2 million which includes US$52 million of preproduction costs.

Table 1-4: Life-of-Mine Capital Costs

Description	US$000's @ $1.05:€1
Initial Capital Costs	
Preproduction Costs	52,003
Mining	69,047
TSF/Process/Infrastructure	403,991
Water Management	10,150
Total Initial Capital	**$535,191**
Sustaining Capital Costs	
Mining	61,208
Process	-
Infrastructure	13,477
TSF	151,282
Water Management	5,154
Total Sustaining Capital	**$231,120**
Total Capital Costs	
Preproduction Costs	52,003
Mining	130,255
TSF/Process/Infrastructure	403,991
Infrastructure (Sustaining)	13,477
TSF (Sustaining)	151,282
Water Management	15,304
Subtotal Capital Costs	**$766,312**
Closure/Reclamation	60,659
Total LoM Capital Costs	**$826,971**

Source: SRK, 2017

Based on an EURUSD of US$1.05:€1.00, Table 1-5 presents total operating costs of US$28.76/t processed used in the Technical Economic Model (TEM).

Table 1-5: Operating Cost Summary

Operating Costs in 000's	@ $1.05:€1
Mining	704,040
Process	621,830
Site G&A	224,309
Water Management	6,368
Total Operating Costs	**$1,556,547**
Operating Cost Unit Rates	**US$/t Proc.**
Mining ($/t mined)	2.44
Mining ($/t processed)	13.01
Process	11.49
Site G&A	4.15
Water Management	0.12
Total Operating Costs	**$28.76**

Source: SRK, 2017

1.15 Economic Analysis

The indicative economic results summarized in this section are based upon work performed by SRK, Lycopodium Minerals Pty Ltd (Lycopodium), or received from Nordgold in 2016. They have been prepared on both an annual pre-tax and after-tax basis, a 100% equity basis with no Project financing inputs, are in Q4 2016 U.S. constant dollars and an EURUSD of US$1.05:€1.00.

The project design is a 4.6 Mt/y operation that would cost an estimated US$535 million of initial capital to build. The project is expected to produce 214,000 oz Au per year at an All-in Sustaining

Cost (AISC) of US$779/oz (including the first 10 years producing 237,000 oz Au per year at an AISC of US$749/oz). Project metrics are summarized in Table 1-6 and show a NPV 5% value of US$370 million and 18.7% Internal Rate of Return (IRR). This valuation is helped in large part by French government surplus tax credit refunds of US$186 million during 2020-2023.

Table 1-6: Project Valuation Summary

Description	US$000's @ $1.05:€1
Net Revenues	$3,058,905
Operating Costs	(1,556,547)
Operating Margin	$1,502,358
Income Taxes (after tax credits)	(200,746)
Operating Cash Flow	$1,301,612
Initial Capital	(535,191)
Sustaining Capital	(231,120)
Closure/Reclamation Capital	(60,659)
Total Capital	($826,971)
Surplus Tax Credit Refunds	185,632
Free Cash Flow	$660,273
NPV 5%	$369,949
IRR	18.7%
Payback from First Production	4.1 years
AISC	US$779/oz

Source: SRK, 2017

Additional gold price sensitivity analyses were performed with after-tax Project NPV 5% and IRR. Table 1-7 shows price sensitivity at a series of discrete price points.

Table 1-7: Sensitivity Analysis at Various Gold Price Points

Gold Price (US$/oz)	NPV@5% (US$ millions)	IRR (%)
971	$0 (Breakeven)	5.0
1,200	307	16.8
1,250 (Base Case)	370	18.7
1,300	433	20.4
1,400	557	23.7
1,500	681	26.7

Source: SRK, 2017

Discount rate sensitivity is important due to the remote location of the Project in a jurisdiction that has little organized mining activity. Discount rate sensitivity in Table 1-8 shows that the after-tax Project NPV positive as currently designed up to an 18.5% discount rate.

Table 1-8: Sensitivity Analysis at Various Discount Rates

Discount Rate	NPV@5% (US$ millions)
0%	660
5% (Base Case)	370
10%	185
15%	63
20%	(19)

Source: SRK, 2017

The Project is also sensitive to the EURUSD exchange rate as operating costs are approximately 77% Euro-based while capital costs are approximately 66% Euro-based. The remaining costs are mainly USD-based. EURUSD exchange rate sensitivity in Table 1-9 shows that the after-tax Project NPV 5% changes approximately US$12 to 13 million for every 100 basis point change in the exchange rate, either upwards or downwards.

Table 1-9: Sensitivity Analysis at Various EURUSD Rates

EURUSD Rate	NPV@5% (US$ millions)	IRR (%)
0.95	497	23.0
1.00	434	20.9
1.05 (Base Case)	370	18.7
1.10	304	16.4
1.16	235	14.0

Source: SRK, 2017

As currently designed and in the current metal price and cost environment, the Project requires significant Overseas Department tax credits to achieve a reasonable return on investment. This situation is highlighted in Table 1-10 which shows the results of the base case which assumes the program would continue through the LoM of the Project past its current 2020 expiry date compared to various levels of tax credit participation. At the extreme, there is a 45% decrease in Project IRR from the base case with full utilization compared to a scenario when they are not used.

Table 1-10: Sensitivity Analysis at Various Tax Credit Levels (US$ millions)

Tax Credit Level	Tax Credits Generated	NPV 5%	IRR (%)	% Var from Base Case IRR
LoM (Base Case)	238	370	18.7	-
5 Yr Extension which ends 2025	207	350	18.2	-2.7
Ends 2020	115	272	14.9	-20.3
Not used	-	166	10.3	-45.0

Source: SRK, 2017

1.16 Conclusions and Recommendations

1.16.1 Geology and Resources

Geology and Resources

- Columbus has completed an industry standard exploration drilling program over an area of approximately 1 1/4 km^2;
- The average drill spacing is approximately 35 m x 50 m in the measured resource, 50 m x 75 m in the indicated resource and 100 m-150 m in the inferred resource'
- The exploration work has been accompanied by an industry standard QA/QC program showing high quality test results'
- Columbus has conducted extensive core logging resulting in a high quality geologic model'
- The results of the drilling, sampling, analytical testing, core logging and geologic interpretation provide good support for an industry standard resource estimation; and

- The results of the Mineral Resource estimation confined within a Whittle™ pit shell optimization, hosts a Measured and Indicated Mineral Resource of 85 Mt at an average Au grade of 1.4 g/t containing 3.9 Moz of gold and an additional Inferred Mineral Resource of 20 Mt at an average Au grade of 1.5 g/t containing 1.0 Moz of gold.

Exploration and Development

- Additional infill drilling at the 35 m x 50 m spacing could be completed in the areas of early mining to provide additional confidence in the tonnes and grade of this production;
- Infill drilling is recommended to target the areas where Inferred Resources are located within the Reserve pit, where the current resource Au block grades are estimated to be above mining CoG. This could in turn convert current Inferred Mineral Resource to Mineral Reserves; and
- Additional sample analysis could also be conducted to refine the current NAG and PAG model.

1.16.2 Geotechnical

The geotechnical field investigation consisted of seventeen drillholes, targeted to characterize rock mass fabric and structural features in and around the mineralized zone at different depths and orientations.

Two major geotechnical domains have been identified in the Project. A hard rock slope composed of strong foliated metamorphic rock and a near surface saprolite soil domain that controls the stability of the upper 30 to 40 m of the ground. The saprolite is a deeply and intensely weathered residual rock that behaves like a soil. It is weak, nearly saturated, and easily deformable. The geotechnical field investigation program was conducted using accepted industry standards and procedures. The data collected is sufficient for a BFS level design. Stability of the overall pit slopes has been demonstrated using industry accepted slope acceptance criteria.

SRK notes that the back analyzed strength results for the saprolite on historic landslides are similar to the 25th percentile distribution strength. Based on the historic saprolite failures, the 25th percentile distribution from the current testing program is appropriate for use in slope stability analysis.

The stability of saprolite slopes has been demonstrated by limit equilibrium calculations. The stability of saprolite slopes is subject to the completion of a drainage design and placement of vegetative cover on all saprolite slopes following excavation. The saprolite slopes will be subject to gullying, erosion, creep, and flow failures if vegetative cover is not established. Mine design parameters have been provided and are appropriate by the state of the practice. SRK recommends that Nordgold implement a slope monitoring program prior to the beginning of mining and earthworks on the Project site.

WRDs should include a coarse underdrain material, which is a minimum of 5 m thick, following the course of any existing drainages. The coarse underdrain may be constructed of Run-of-Mine (RoM) waste.

1.16.3 Mining and Reserves

The mine block model, geotechnical stability, pit design, phase design, dump design, production schedule and reserve estimation have been completed to BFS level criteria. The Project confirms a positive cash flow using only Measured and Indicated resources for the conversion of reserves using a USUS1,200/oz gold price. The mine design supports the style and size of equipment selected for operations with weather corrections applied to various months of the year accounting for the tropical and potentially wet periods of time. While subject to continual improvement, the mine plan implementation will require qualified staff and the integration of all mining and related disciplines for the successful execution of the Project.

Additional mining related studies for a detailed engineering level of design for the Project include:

- Detailed scheduling for pre-production earthworks;
- Continued discussion with vendors for equipment quotes;
- For detailed engineering, the low-grade saprolite stockpile design should be advanced;
- Development of operational guidelines for treatment of ARDML waste rock. Customization of rapid PAG field testing would also be advized; and
- An infill drilling program to optimize mine design related to the pit toe of the reserve pit, internal waste intrusions, saprolite/hardrock interface and grade variability.

1.16.4 Mineral Processing and Metallurgical Testing

The following significant factors are identified based on the metallurgical studies conducted for the Montagne d'Or BFS:

- The BFS metallurgical program focused on the development of a process flowsheet that included gravity concentration followed by cyanidation of the gravity tailings and intensive cyanide leaching of the gravity concentrate;
- Montagne d'Or ore can be readily processed to recover the contained gold and silver values using unit operations considered standard to the industry; and
- SRK has estimated overall adjusted gold and silver recoveries based on the contribution from each ore lithology during each phase of mining. During the first six mining phases gold recovery is estimated at 94% to 95% and silver recovery is estimated at about 54% to 56%. These recovery projections include a 2% deduction from reported laboratory test results to account for inherent plant inefficiencies.

Detoxification of the cyanide leach residues was accomplished with the industry-standard SO2/Air process. It was demonstrated that cyanide in the leach residue could readily be detoxified to less than 1 ppm CN_{wad}. SO_2 consumption in the range of about 5 to 6 g SO_2/g CN_{wad} were reported, which is typical of industry practice.

1.16.5 Recovery Methods

The process plant will be designed for a nominal 4.6 Mt/y throughput on the design ore blend of 89% felsic tuff, 7% granodiorite and 4% mafic. The design will allow the nominal throughput to be achieved in 8,000 operating hours per year.

The flowsheet comprises a primary jaw crusher feeding a surge bin with excess crushed ore reporting to a dead stockpile which can be reclaimed by FEL to the surge bin. A single stage semi-autogenous grinding circuit with recycle crushing (SS SAC) comminution circuit with a 14 million watt (MW) SAG mill, recycle pebble crusher and cyclones will produce a target P_{80} 75 µm grind size. A gravity circuit will recover coarse gold. The milled slurry will be thickened prior to reporting to a standard CIL leach circuit proving a total of 31 hours residence time. A split AARL 10 t elution circuit will recover the gold for electrowinning. The leach tails will be diluted with incoming plant make-up water prior to thickening to reduce the contained cyanide prior to reporting to the cyanide destruction circuit using the SO_2/air technology to ensure plant tailings comply with the environmental requirements. The plant tailings will be pumped to the TSF with decant return from the tailings embankment returned to the plant as make-up water.

A moderate level of automation and remote control will be provided to ensure safe operation of the plant and to control process conditions for optimum recovery. Operators will monitor the plant to ensure that spillage is detected and cleaned up quickly and that good housekeeping practices are followed in compliance with safe working practices and country regulations.

Additional field investigations are required prior to final plant foundation design. Drilling complemented by Standard Penetration Tests (SPTs) and cone penetration tests (CPTs) is recommended to confirm foundation conditions for final design. The additional field work should consist of 20 to 30 holes with SPT logging and, where appropriate, CPT probes located within the foundation footprint. The number of drillholes may be reduced if geophysical surveys of the saprolite, saprock and bedrock contacts can be successfully completed. Additionally, a geophysical investigation should be conducted to determine dynamic soil properties. Additional geotechnical characterization, laboratory and field testing of the saprolite soils, and the potential need for planned additives such as waste rock and/or lime should to be conducted to provide data to in bring cost estimates to a final design level.

1.16.6 Project Infrastructure

Project development is dependent on the access road between SLM and Camp Citron via Apatou Crossing. Although the portion of the road between SLM and Apatou Crossing is currently suitable for all-weather traffic the 54 km of road from Apatou Crossing to Camp Citron is not.

It is recommended that consideration be given to the early commitment of funds to partially upgrade the Apatou Crossing/Camp Citron road during the dry season prior to Project commencement to facilitate mobilisation of personnel and materials to site for early activities such as sediment control earthworks, forest clearance and establishment of pioneer accommodation, office, communications, contractor facilities and fuel supplies.

The estimated capital cost for the complete road upgrade is just under US30 million. A commitment of some 25% of this sum would secure a much improved level of road access to the site during the 2018/2019 wet season.

It should be noted that the capacity of the French Guiana power network to support the Project power demand has not been rigorously tested as supply is dependent on a yet to be constructed power station in the north west of French Guiana.

During the planning and permitting stages a firm commitment for the provision of an adequate power supply at the point of tie-in must be obtained before the high voltage infrastructure can be designed and commitment made to constructing the overhead power line.

1.16.7 Tailings Storage Facility

The principal objective of SRK's TSF design was to ensure protection of the regional groundwater and surface waters during operations and closure, while containing solid waste materials within an engineered facility, provide a design that is geotechnically stable over four phases to minimize initial capital, and meet closure objectives. SRK developed a phased TSF design which contains approximately 56 Mt of tailings, corresponding to approximately 12 years at a rate of 4.6 Mt/y, and follows the French Guiana requirements for BAT.

SRK recommends that the following key items be considered in further detail in order to reduce the uncertainty associated with the BFS TSF engineering design:

- A detailed Project schedule should be developed that considers the contractor equipment, earthwork quantities (including wastage) and dry/wet seasons;
- Additional field investigations should be performed in the TSF footprint areas, including characterizing the foundation conditions, tailings material, and potential borrow areas;
- The anticipated tailings supernatant water quality should be reviewed and the assumption that supernatant can be recirculated through the process and the tailings area net AGP should be confirmed; and
- The closure design should be reviewed and confirmed during the detailed design based on permitting discussions.

1.16.8 Site Water Management

Hydrogeology

Approximately 40% of the predicted total inflow into the open pit is coming from groundwater and the remaining water is sourced from direct precipitation and run-off. The primary sources of groundwater inflow to the pit are 1) captured groundwater in saprock that discharges to the pit from the south highwall, and 2) depletion of groundwater storage.

When mining ceases, the open pit will fill with a combination of groundwater and a predominant amount of run-off and direct precipitation. The initial groundwater contribution will be about 40 % of the total inflow. Groundwater inflow will decrease as the lake fills, and will comprise a small component of inflow once the pit lake reaches the overflow point. Once the pit lake reaches the overflow point of the pit, it will be captured and routed to undisturbed drainages, because the pit lake water quality is expected to be suitable for discharge.

The following recommendations are appropriate for assessing long term impacts and for monitoring water levels as mining begins:

- Continue the creek flow accretion monitoring on Apollon and Infirmes creeks. Analyze the data acquired between August of 2016 through August of 2017 on a continuous basis and make interpretations on baseflow in the creeks.
- After a set of flow accretion data from a full dry season has been analyzed, recalibrate the numerical model to observed baseflows. Reassess impacts to the high creeks in the RBI, and specifically Apollon Creek.
- Prior to mining, add a set of 3 nested piezometers or observation wells above the pit perimeter, and along the full extent of the pit rim. Complete either nested vibrating wire piezometers or nested standpipe wells in the saprock and bedrock, respectively, at 3

locations. These installations will allow Nordgold to assess the materials above the pit for geotechnical stability. Furthermore, they will allow Nordgold to track dewatering progression above the pit, between the mining operation and the RBI.

Site Water Management

Site water management at the Project includes management of stormwater run-off at the site and the management of the accumulation and consumption of contact and process water within the mine facilities.

Stormwater is addressed by diverting run-on to the Project around the facilities so that it remains non-contact water. The non-contact diversion system includes almost 15 km of ditches, road side channels, and diversions around the WRDs, pits, stockpiles, and TSF. Some of these diversions will be covered as the Project facilities expand and will be reconstructed as needed in response to the facility growth. In addition, seven sediment control ponds have been located around the Project, downstream of the diversions to collect and control sediment laden waters released from the site to prevent non-compliant sediment releases.

SRK developed a detailed water balance model of the Project, tracking the movement of water and select solids (i.e. ore, tailings, and waste rock) within the Project.

Contact water generated by the Project will be routed to the CWP where it can be stored for future use as Project makeup. However, the water balance modeling indicates the system will consistently run positive and excess contact water must be discharged from the system in order to prevent uncontrolled releases. Water balance modeling indicated that treatment and discharge from the system at a rate of up to 180 L/sec is required to maintain a net neutral or net negative balance in the system. The design capacity of the CWP allows sufficient water to be stored to provide makeup to the process even in times of extreme drought, yet maintain sufficient surge capacity to prevent uncontrolled releases during periods of high rainfall.

The water balance model assumed that once the closure cover was completed on the TSF, clean, non-contact water could be discharged from the TSF to the environment and treatment of the TSF waters would no longer be required.

From a hydrology standpoint, the site as a great capacity to produce high volumes of run-off that can have significant impact on mining activities. The following recommendations are provided to increase the understanding of the site hydrology and the management of water at the mine site;

- Management of the TSF supernatant pool is limited to a narrow range during operations, with the intent of maximizing the area of exposed beach to enhance consolidation, and to provide a large surge capacity to contain the inflow from extreme storm events. Maintaining such tight control will require diligent monitoring of the TSF pool and establishing of reliable method of predicting inflows. The system should be prepared to address the possibility of high rainfall at any time during operations that will result in unexpected inflows to the TSF water management system;
- Similarly, the Contact Water Management system must maintain a delicate balance between ensuring sufficient water is available to sustain operations during an extreme drought, while at the same time maintaining sufficient surge capacity within the CWP to contain the inflow from extreme storm events. Criteria by which the pool is managed, begun in this study, must be expanded as the understanding of the Project expands;

- Design elements for the Sedimentation Ponds and the CWP will need to be included. Design elements to include intake and outlet control structures, erosion management, excavation and grading. Designs are required prior to finalizing the position of the water management diversion ditches and energy dissipation structures. Detailed engineering of the mine water management components will be required to advance this Project to design level;
- SRK is aware of continued climate and streamflow monitoring at the site. This data should be used to regularly update the understanding of the climatic conditions and hydrological behavior at the site. Refinement of these behaviors could have significant ramifications on mine water management at the site; and
- The tropical environment at the Project will necessitate regular maintenance of all diversion ditches and sediment ponds.

Geochemistry

Findings, conclusions, and recommendations from the geochemical test data and predictive modeling are summarized below. Details of the geochemical testing program on waste rock and tailings along with results and interpretations are described in SRK (2017a). Descriptions and results of the predictive geochemical models are presented in SRK (2017b).

Tailings:

- Tailings pore water will be alkaline (pH 8.5) when first discharged, and concentrations of metals and cyanide will be below regulatory limits; however the capacity of the alkalinity in the tailings to neutralize acid drainage is low;
- Tailings solids will carry a sulphide concentration of 1.2% (primarily pyrite with subordinate chalcopyrite and pyrrhotite), and negligible acid neutralizing carbonate minerals, resulting in a strongly net acid generating condition;
- Tailings could be acid generating in the beaches and embankment locations that are subjected to intermittent wetting and drying; and
- Tailings that are in a fully submerged condition should maintain circum-neutral pH with metal and cyanide concentrations below regulatory limits.

Waste Rock:

- Felsic tuff and lapilli tuff deposited in WRDs are predicted to be net acid generating, amounting to 41% of waste rock categorized as PAG. The other primary rock types (saprolite, saprock, felsic porphyry, granodiorite, quartz feldspar porphyry, mafic volcanics, and diabase dikes) are expected to be net acid neutralizing;
- Due to the excess volume of non-PAG rock deposited on WRDs in early years, dump drainage pH is predicted to be >5 in years -2 and -1 (base case without cover);
- As the relative volume of PAG rock increases disproportionately to non-PAG rock through year 3, waste rock drainage pH decreases and will be sustained at 3 – 3.5 until the end of mining, and copper exceeds regulatory limits (base case without cover); and
- A closure strategy of cover emplacement concurrent with waste rock deposition, in conjunction with a material handling and segregation plan, could significantly attenuate the production of acid rock drainage from waste rock

Pit Lake:

- Based on groundwater model calculations, the pit lake will refill rapidly and is predicted to attain maximum depth and overtop 73 months after the start of infilling (base case without contact water pumpback);
- The rapid infilling results in substantial dilution that is predicted to minimize sulphide oxidation and flushing of weathering products into the lake; and
- Throughout all infilling stages and into closure, the pit lake pH is predicted to be sustained around 8.1, with all metals concentrations below regulatory limits.

1.16.9 Environmental Studies and Permitting

From and environmental and permitting perspective, the most important issues centre around the accurate characterization of AGP of the various geological materials, and the proper management and disposal of those materials once excavated from the open pit. SRK recommends that a detailed mine schedule be developed using the geological block model that is based on the ARDML potential of the rock so that the deposition of these materials can be sequenced within the WRDs in a manner that places inert materials on the exterior of the facility, while sequestering potentially reactive materials in the interior. This will minimize the surface exposure of sulphidic materials to oxygen and precipitation, and allow for more effective management and closure of the WRDs, thus reducing the need for longer-term seepage monitoring and collection.

Along the same lines, SRK recommends that a complete site-wide inventory of all potential closure cover materials be performed; that geochemical, geotechnical, and agronomical testing of these materials be conducted; and that infiltration modeling of potential cover design be completed. This will allow Nordgold to move away from the prescriptive, regulatory cover designs to more practical designs that can demonstrate equal or better protection of the environment post closure.

Finally, addition baseline data collection will likely be required on the mineral concession, Concession 102 ("01/32"), on which the proposed TSF is partially located, which is not currently owned by Nordgold.

1.16.10 Recommended Work Programs and Costs

As provided by Nordgold, there exists budgeted spending of approximately US$2 million per year for 2017 and 2018 for management, environmental permitting and ongoing operations including:

- Project management;
- Regulatory and environmental specialists and consultants;
- In-country office costs;
- Public relations, community relations and stakeholder engagement programs; and
- Administration and other overheads.

(For the purposes of the BFS the budgeted costs of US$2 million per year for 2017 and 2018 were considered to be sunk costs, and were not included in the Project capital costs.)

Geology and Resources

At this time, the current drilling and resource estimate is sufficient for further advancement of the Project up to point of making a go-ahead decision. Infill drilling is recommended to target the areas where Inferred Resources are located within the Reserve pit where the current resource Au block grades are estimated to be above mining CoG. This could in turn convert current Inferred Mineral Resource to Mineral Reserves.

Plant Site Geotechnical

SRK recommends completing a final geotechnical design for the plant site. The following studies and parameters should be completed and appropriate design values verified:

- A soil geophysical survey of the site should be completed to establish the bedrock depth and determine dynamic properties, including the dynamic shear modulus. This survey can also be used to determine the depth to bedrock across the plant foundation for dimensioning of pile foundations;
- CPT or SPT drilling and testing should be completed at the final foundation locations to verify soil conditions used in this analysis and to complete a final design. This is recommended as a soils rig including SPTs was not available for this program; and
- Additional testing should be completed for characterization including ASTM D4647 (pinhole test) and soil resistivity.

The cost estimate for these programs is US$130,000.

Mining and Reserves

Drilling recommendations previously mentioned are optional. Other work recommendations would be carried out as part of normal detailed engineering, procurement and construction management (EPCM), or as part of mining engineering work during the pre-production mining period. Therefore, associated costs for mining related programs would be already included in normal detailed engineering costs and pre-production mining costs. There are no additional costs required for the Project at this stage prior to a decision to go into construction.

Mineral Processing and Metallurgical Testing

Metallurgical testing performed to date is sufficient for advancement of the Project up to and including a decision to construct the Project.

Recovery

There are no recommended work programs required prior to a decision to construct the Project.

Project Infrastructure

There are no recommended work programs required prior to a decision to construct the Project.

Tailings Storage Facility

SRK recommends the following work be performed prior to the construction of the starter earthworks and the commencement of operations:

- Prior to the development of construction drawing and specifications, additional field investigations should be performed in the TSF footprint areas, including complementary characterization of the foundation conditions (i.e. where significant gaps exist), tailings material, and potential borrow areas, with an estimated cost of US$400,000;
- A field and laboratory program should be performed to characterize the in situ permeability and attenuation characteristics of the underlying saprolitic soils, as well as potential

permeability amendment options for the TSF foundation soils. This data would be used to support a numerical groundwater model and demonstrate compliance with French regulations. If combined with the TSF foundation characterization program, it has an estimated additional cost of between US$100,000 to US$250,000;

- Prior to the development of construction, TSF final design drawing and specifications should be completed, which are part of the planned BFS engineering budget (subsequent to a decision to construct the Project); and
- An OMS manual which document operations, monitoring and surveillance should be developed, and is part of the planned BFS engineering budget (subsequent to a decision to construct the Project).

Site Water Management

Recommended hydrogeology, hydrologic and climatological study costs would be covered by the planned Project permitting budget by Nordgold (for regulatory and environmental specialists), and that there are no additional costs anticipated.

Detailed engineering of the mine water management components will be required to advance the Project to design level. However, these are included as part of the planned engineering budget subsequent to a decision to construct the Project and there are no additional recommended work program costs prior to a decision to construct the Project.

Geochemistry

SRK has previously noted that a series of long-term column leach tests would supplement the geochemical data obtained for the WRDs. This program is not critical for the next phase of the Project, and is not necessary for making a decision to proceed with construction of the Project. The recommended soil attenuation program for the WRD foundations is discussed in the Environmental section below.

Environmental

Recommendations regarding material excavation and sequenced disposal would be carried out as part of normal detailed engineering of the WRDs, or as part of mining engineering work during the pre-production mining period. Therefore, associated planned costs would be mainly included already in normal detailed engineering costs and pre-production mining costs.

The identification, sampling, and characterization of closure cover materials is dependent on the number of sources investigated, and could be deferred to the end of mining if not desired for initial permitting. Depending on the interpretation by French regulators, there is a risk that a materials investigation will be needed to confirm the quality and quantity of these materials.

Additional baseline data collection will be limited to the encroachment footprint of the TSF onto Concession 102, and could likely be covered by normal operating costs associated with the ongoing permitting efforts.

SRK also recommends that a field investigation should be performed within the proposed WRDs, CWP and LG ore stockpile footprint areas, to characterize the in situ permeability of the underlying saprolitic soils, foundation characteristics, and potential permeability amendment options for the foundation soils, with an estimated cost of between US$225,000 to US$625,000. In addition, materials collected during this field program would be subjected to attenuation testing with the objective of demonstrating the effectiveness of chemical constituent removal from seepage contacting and passing through the barrier systems. The chemical attenuation program has an estimated cost of between US$125,000 and US$250,000.

Capital and Operating Costs

Although no Value Added Tax (VAT) is applicable in French Guiana (by exception to the other French overseas districts), the following French Guiana import taxes should be anticipated unless specific measures will be been granted to Project. In SRK's view these taxes do get commonly waived for mining projects in many jurisdictions so the subsequent risk is low but these taxes include:

- Customs duties: goods imported from third countries (outside the EU) are potentially submitted to customs duties, depending on their origin from a custom viewpoint. The rate will depend on the nature of the assets as determined by the customs tariff;
- External dock duties ("octroi de mer externe"): The import dock duties are due when goods (inventories or fixed assets) are imported in French Guiana from any other territory (Metropolitan France, other French overseas districts, European Union (EU) Member States or third countries). They are assessed on the purchase price plus custom duties. The rate could range between 0% (many exemptions applicable) and 60 %, depending on the tariff. With respect to this case, it is anticipated that most of the assets should be subject to rates of 7.5%, 15% or 22.5%, plus a regional 2.5% duty;
- Internal dock duties ("octroi de mer interne"): the sale of products manufactured, transformed or extracted locally is submitted to internal dock duties, with the same rates. However, a producer submitted to the internal dock duties has a right to deduct external dock duties suffered for its production, especially when the good produced are exported. As a result only the value added is consequently submitted to the internal dock duties in such case;
- The depreciation basis of the imported assets should include both customs duties and external dock duty if not recoverable under the conditions explained above; and
- An import duty review program is recommended at a cost of US$20,000.

With respect to labour costs in French Guiana, SRK recommends the resolution of the issue to identify the impact of the benefit of some social security exemption according to a specific oversea regulation (LODEOM Renforcée) or the general French social security exemption (reduction FILLON). For the purposes of this study, Nordgold retained the less favourable scenario as it is not guaranteed that you could benefit from both the "LODEOM Renforcé" scheme up to 250 employees and the FILLON scheme for the remaining eligible employees or to obtain the benefit of the LODEOM Renforcé for all employees. A labour regulation review program is recommended at a cost of US$20,000.

Technical Economics

SRK recommends that the French Overseas Department tax credit program be evaluated in further detail due to the importance of the surplus tax credit refunds in the early part of the mine life. In particular, it would be useful to receive more information about the eligibility of preproduction costs, the TSF and the water management costs in the calculation of for the tax credit. Also, given the size of the Project, it is certain that the tax credit will be subject to a prior approval to be given in advance by the French Central Tax Authorities. A tax credit review program is recommended at a cost of US$15,000.

LoM long range EURUSD exchange rate forecast surveys should be done as the exchange rate has a strong impact on Project economic metrics. An exchange rate forecast program is recommended at a cost of US$15,000.

1.16.11 Summary of Recommended Work Program Costs

Recommended work program costs are summarized in Table 1-11.

Table 1-11: Summary of Costs for Recommended Work

Recommended Work Programs	Cost Estimate (US$)
In-fill Drilling on Inferred Resources within Reserve Pit	350,000
Plant Site Foundations Geotechnical Programs	130,000
WRDs/LG Stockpile Foundation Characterization Program	225,000 to 625,000
Soil Attenuation Investigation	125,000 to 250,000
TSF Geotechnical Characterization and Groundwater Modeling Program	500,000 to 650,000
Import Duty Review Program	20,000
Labour Regulation Review Program	20,000
Tax Credit Review Program	15,000
Exchange Rate Forecast Program	15,000
Total Programs	**$1,400,000 to $2,075,000**

Source: SRK, 2017

2 Introduction

2.1 Terms of Reference and Purpose of the Report

This report was prepared as a feasibility level NI 43-101 Technical Report for Nordgold with Columbus by SRK on the Montagne d'Or Project, located in the commune of SLM, French Guiana. Columbus is the Project owner and is currently exploring the deposit under an option agreement with Nordgold, the Project operator.

The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in SRK's services, based on: i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. This report is intended for use by Nordgold subject to the terms and conditions of its contract with SRK and relevant securities legislation. The contract permits Nordgold and Columbus to file this report as a Technical Report with Canadian securities regulatory authorities pursuant to NI 43-101, Standards of Disclosure for Mineral Projects. Except for the purposes legislated under provincial securities law, any other uses of this report by any third party is at that party's sole risk. The responsibility for this disclosure remains with Nordgold. The user of this document should ensure that this is the most recent Technical Report for the property as it is not valid if a new Technical Report has been issued.

This report provides Mineral Resource and Mineral Reserve estimates, and a classification of resources and reserves prepared in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum Standards on Mineral Resources and Reserves: Definitions and Guidelines, May 10, 2014 (CIM, 2014).

2.2 Qualifications of Consultants

The Consultants preparing this technical report are specialists in the fields of geology, exploration, Mineral Resource and Mineral Reserve estimation and classification, open pit mining, geotechnical, environmental, permitting, metallurgical testing, mineral processing, processing design, capital and operating cost estimation, and mineral economics.

None of the Consultants or any associates employed in the preparation of this report has any beneficial interest in Nordgold or Columbus. The Consultants are not insiders, associates, or affiliates of Nordgold. The results of this Technical Report are not dependent upon any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings between Nordgold and the Consultants. The Consultants are being paid a fee for their work in accordance with normal professional consulting practice.

The following individuals, by virtue of their education, experience and professional association, are considered Qualified Persons (QP) as defined in the NI 43-101 standard, for this report, and are members in good standing of appropriate professional institutions. QP certificates of authors are provided in Appendix A.

The QP's are responsible for specific sections as follows:

- Bart Stryhas, SRK Principal Resource Geologist, is the QP responsible for background, geology and resource estimation Sections 1.1-1.4, 1.16.1, 4 except 4.3 and 4.4, 6-12, 14, 23, 25.1, 26.1 of this Technical Report;
- John Tinucci, SRK Practice Leader/Principal Consultant (Geotechnical), is the QP responsible for geotechnical Sections 1.5, 1.16.2, 16.1, 25.2, 26.2 of this Technical Report;
- Bret Swanson, SRK Practice Leader/Principal Consultant (Mining Engineer), is the QP responsible for mine design and mine planning Sections 1.6, 1.7.1, 1.16.3 (shared), 15, 16.2, 16.3, 25.3, 25.4 (shared), 26.3 of this Technical Report;
- Peter Clarke, SRK Principal Consultant (Mining), is the QP responsible for mining Sections 1.7.2, 1.16.3 (shared), 2, 3, 5.1, 5.3, 5.5, 5.6 (shared), 5.7 (shared), 16.4, 25.4 (shared), 26.3 (shared), 26.12, 27 and 28 of this Technical Report; and
- Eric Olin, SRK Principal Consultant (Metallurgy), is the QP responsible for mineral processing and metallurgy Sections 1.8, 1.16.4, 13, 25.5, 26.4 of this Technical Report.
- David Bird, SRK Principal Consultant (Geochemistry), is the QP responsible for geochemical testing of tailings and waste rock, interpretation of data, and predictive geochemical modeling of tailings, waste rock, and pit lake Sections 1.12.3 and 26.8.2 of this Technical Report;
- Paul Williams, SRK Principal Consultant (Hydrogeology), is the QP responsible for hydrogeology Sections 1.12.1, 1.16.8 (shared) and 16.1.9 (shared), 16.5 (shared), and 26.8.1 of this Technical Report;
- David Hoekstra, SRK Principal Consultant (Civil Engineer), hydrology Sections 1.12.2, 1.16.8 (shared), 16.1.9 (shared), 16.5 (shared), 18.2.3, 25.9, 26.8.3 of this Technical Report;
- Cameron Scott, SRK Principal Consultant (Geotechnical Engineer), is the QP responsible for TSF Sections 1.11, 1.16.7, 18.3, 25.8, 26.7 of this Technical Report;
- David Gordon, Lycopodium Manager of Process, is the QP for process, recovery and infrastructure Sections 1.9, 1.10, 1.14 (shared), 1.16.5, 1.16.6, 5.2, 5.4, 5.6 (shared), 5.7 (shared), 17, 18.1, 18.2.1, 18.2.2, 21 (shared), 24, 25.6, 25.7, 26.5, and 26.6 of this Technical Report.
- Mark A. Willow, SRK Principal Environmental Scientist/Practice Leader, is the QP responsible for environmental studies, permitting and social or community impact Sections 1.13, 1.16.9, 4.4, 20, 25.10, 26.9 of this Technical Report;
- Grant A. Malensek, SRK Principal Consultant (Mineral Economics) is the QP responsible for economics and market Sections 1.14 (shared), 1.15, 1.16.10, 1.16.11, 4.3, 19, 21 (shared), 22, 25.11, 25.12, 26.10 and 26.11 of this Technical Report;

2.3 Details of Inspection

Bart Stryhas, Bret Swanson and Mark Willow visited the Project site for three days on April 1-3, 2014. Additionally, Bart Stryhas, Peter Clarke, Cameron Scott, John Tinucci, Paul Williams, David Bird and Bret Swanson visited the site for two days each, between October 12-17, 2015. During the various site visits, the group toured the general areas of mineralization, historic mining, drilling sites, reviewed existing infrastructure, observed the Columbus drill core and reviewed Project data files with Columbus' and Nordgold's technical staff.

Chris Waller of Lycopodium visited French Guiana for five days between October 13-18, 2015 including visiting the site for two days from October 16-17, 2015 to review existing site infrastructure, view potential sites for the process plant and infrastructure and to evaluate the condition of the site end of the access road. Steve Evans and Luciano Giancristofaro of Lycopodium visited French Guiana between February 28 and March 9, 2016, met with local contractors, visited the ports of Cayenne and Saint Laurent du Maroni, drove the road from Cayenne to Apatou Crossing and visited the Project site. Dave Gordon (QP, Process) of Lycopodium did not conduct a site visit, as sufficient site investigations were conducted by other Lycopodium personnel.

A site visit was not conducted by David Hoekstra (QP, Hydrology) of SRK, as Paul Williams (QP, Hydrogeology) of SRK conducted two site visits collecting sufficient data and information.

It was not necessary for Eric Olin (QP, Metallurgy) and Grant Malensek (QP, Economics) of SRK to conduct site visits.

Table 2-1: Site Visit Participants

Personnel	Company	Expertise	Date(s) of Visit	Details of Inspection
Bart Stryhas	SRK	Geology/Resources	April 1-3, 2014, October 12-17, 2015	Drill Core/ Field Geology
Bret Swanson	SRK	Mining	April 1-3, 2014	Project area
Mark Willow	SRK	Environmental	April 1-3, 2014	Project area
John Tinucci	SRK	Geotechnical	October 12-13, 2015	Project area/ Drill Core
Paul Williams	SRK	Hydrology	October 12-13, 2015, June 28-29, 2016	Project area
David Bird	SRK	Geochemistry	October 12-13, 2015, November 10-11, 2015	Project area
Bret Swanson	SRK	Mining	October 12-13, 2015	Project area
Bart Stryhas	SRK	Geology Resources	October 16-17, 2015	Drill Core/ Field Geology
Peter Clarke	SRK	Project Manager	October 16-17, 2015	Project area
Cameron Scott	SRK	Tailings	October 16-17, 2015	Project area
Chris Waller	Lycopodium	Process & Infrastructure	October 13 –15 2015	Cayenne
Chris Waller	Lycopodium	Process & Infrastructure	October 16-17 2015	Project area
Steve Evans	Lycopodium	Roads & Infrastructure	February 28 – March 9, 2016	Ports, roads, site
Luciano Giancristofaro	Lycopodium	Logistics and Construction Management	February 28 – March 9, 2016	Ports, roads, site

Source: SRK, 2015

2.4 Sources of Information

The sources of information include data and reports supplied by Columbus and Nordgold personnel as well as documents cited throughout the report and referenced in Section 27. The electronic database was compiled and transmitted by Columbus.

2.5 Effective Date

The effective date of this report is March 6, 2017.

2.6 Units of Measure

The metric system has been used throughout this report. Tonnes are dry metric of 1,000 kilograms (kg), or 2,204.6 lb. All currency is in US Dollars (US$ or USD) except where Canadian Dollars (C$) and Euros (€) are stated. The Euro:USD exchange rate (EURUSD) used for estimating the original base costs in this report was US$1.10:€1.00. Subsequently, in the technical economic model for the Project these cost estimates have been converted based on an EURUSD of US$1.05:€1.00 to better reflect long range forecasts. Economic modeling has also been performed using an EURUSD of US$1.05:€1.00, so that all final financial metrics are based on that same rate.

3 Reliance on Other Experts

The Consultant's opinion contained herein is based on information provided to the Consultants by Columbus throughout the course of the investigations. SRK has relied upon the work of other consultants in certain Project areas in support of this Technical Report.

SRK has relied on Columbus's legal representation to describe the:

- Geopolitical;
- Mineral Rights;
- Nature and Extent of Ownership, and
- Royalties, Agreements and Encumbrances.

These items have not been independently reviewed by SRK, and SRK did not seek an independent legal opinion of these items.

SRK has relied on the FIDAL law firm (France) for information to address various Project financial aspects including:

- Information based on the standard French corporate income tax (CIT) regime regardless of the potential tax advantages that can be granted within the framework of the concession/specific agreement concluded by the State with Nordgold/Columbus;
- French tax laws;
- Tax credits;
- Carry forward losses; and
- Depreciation methods and eligible assets.

Portions of text included in Section 4 have been reviewed by Columbus staff. The Consultants used their experience to determine if the information from other reports was suitable for inclusion in this Technical Report and adjusted information that required amending. This report includes technical information, which required subsequent calculations to derive subtotals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, the Consultants do not consider them to be material.

4 Property Description and Location

Montagne d'Or is located along the northern flank of the Dékou-Dékou range. Montagne d'Or (Paul Isnard sector), consists of eight mining concessions and two exclusive exploration permits (PER) located in the commune of Saint-Laurent-du-Maroni, NW French Guiana. The Project also includes historic artisanal mining operations, exploration roads and drill pads, a core logging/storage facility and Camp Citron. The camp hosts a main core shack/office building and approximately six bunkhouse/shower buildings.

4.1 Property Location

The Project area, consisting of a total of 18,949 hectares, and mining concessions are located in the northwestern portion of French Guiana, South America (Figure 4-1). The Project area extends from longitude 53° 53' 52'' W (UTM 178,475) to 54° 03' 09'' W (UTM 161,360), and latitude 4° 40' 59'' N (UTM 518,322) to 4° 51' 03'' N (UTM 536,922). Camp Citron, the base camp for the Project, is located approximately 4 km northwest of the deposit. All Project coordinates are referenced to the "Official" French Guianese Reseau Geodesique Francais (RGF) G95 Zone 22N UTM coordinate system.

Source: Columbus, 2015

Figure 4-1: Montagne d'Or Project General Location Map

4.2 Mineral Titles

4.2.1 Geopolitical

French Guiana is governed by the provisions of the French Constitution as a territorial collectivity of France ("CTG") and, as such, forms an integral part of the French Republic. It sends two elected representatives to the National Assembly and two to the Senate. Local governance is headed by a Prefect and by a 51-member Assembly whose members are elected by universal adult suffrage.

SDOM Mining Legislation

The President of the French Republic, Mr. Sarkozy (at the time of legislation), committed himself to a new comprehensive mining legislation in French Guiana following his rejection, in February 2008, of IAMGOLD's development application for the Camp Caiman gold deposit. The mining Project demonstrated the difficulties and contradictions related to the compatibility of industrial development and the protection of the environment in the Department.

The new mining legislation, referred to as the *Schéma Départemental D'Orientation Minière de la Guyane* (SDOM), was drafted by representatives of the national government of France in the Prefecture of French Guiana following broad consultation with regional communities, the economic players concerned, environmental protection organizations, trade unions, the State and local and regional bodies competent in the fields of natural and human environment, biodiversity and geology. The final SDOM legislation was approved by decree (*décret n^{o} 2011-2106*) on December 30, 2011, by the *Conseil d'État* (State Council), the highest administrative court in France, and went into effect on January 1, 2012.

The legislation was created with the dual objectives of encouraging economic development of the mining industry in French Guiana while protecting its environment and provides incentive, including security of land tenure and clear guidelines to mining development and environmental conditions and restrictions, to serious and environmentally responsible mining companies while inhibiting environmentally damaging illegal mining activities.

Under the SDOM legislation, the territory of French Guiana is divided into four land use classifications, defined as Zones 0, 1, 2 and 3 (the SDOM Zones), that clearly outline areas where the possibility of prospecting and mining are defined in accordance with Article L.621-1 of the *code minier* (Mining Code). The classification takes into consideration the necessity to protect sensitive natural environments, landscapes, sites and populations, a balanced management of the land and the natural resources, economic interests, and sustainable development of the mining resources, within the limits of current knowledge of the biodiversity and the mineral wealth. The areas where mining activity are permitted represents 55% of the territory:

- Zone 0: Banned for prospecting and mining.
- Zone 1: Open to airborne surveys, underground mining authorized subject to conditions.
- Zone 2: Open to prospecting, underground and open pit mining authorized subject to conditions.
- Zone 3: Open to prospecting and underground and open pit mining.

The Montagne d'Or gold deposit is located within an area classified as a favourable zonation (Zone 2), where all prospecting and mining activity is authorized, although subject to conditions as it lies in proximity to the Lucifer Dékou-Dékou biological reserves (RBI LDD).

Conditions to mining in Zone 2, which in actual fact would be applicable to large scale commercial mining operations anywhere in French Guiana include:

- Demonstration of a viable mineral deposit;
- Adherence to a Charter of Good Practices approved by the State representatives;
- Completion of an Environmental Impact Study and Reclamation Plan; and
- Requirements in Zone 2 can include additional reclamation or environmental investigations as may be required for the public interest, on or off site.

Lucifer and Dékou-Dékou Biological Reserve

The initial RBI LDD was created in 1995 over an area covering 110,300 hectares.

Following the implementation of the SDOM legislation, an Order by the Ministry of *l'écologie, du développement durable et de l'énergie* (MEDDE) and the Ministry of *l'agriculture, de l'agroalimentaire et de la forêt* (Agreste), referred to as the '*Arrêté du 27 juillet 2012'*, was issued in July, 2012, to create and establish the boundaries of the RBI LDD. The biological reserve covers 64,373 hectares and is administered by the ONF.

The principal objectives of the biological reserve are to permit the evolution of the natural forest ecosystem, the preservation of biological diversity and to improve scientific knowledge on the Lucifer and the Dékou-Dékou massifs. To attain these goals human activities within the biological reserve are regulated and logging, prospecting and mining are prohibited.

The RBI LDD is separated into two domains located immediately north and east and south of Montagne d'Or concessions, referred to as Lucifer and Dékou-Dékou, respectively.

To the south of the Montagne d'Or Mineral Resource, the boundary of the Dékou-Dékou portion of the biological reserve is defined from west to east by:

- The 420 m elevation line over a distance of 5.5 km;
- A 0.8 km straight line oriented 107^{o} azimuth starting at the 420 m elevation extending to the 505 m elevation and then rejoining the 420 m elevation; and
- Extending southeast along the Apollon creek bed over a distance of 2.8 km.

The location of the Dékou-Dékou biological reserve with respect to the pit outline is shown in Figure 4-2. There is currently a minimum 440 m set-back between the reserve boundary and the BFS pit outline.

The southern portion of the concession C02/46 that falls within the RBI LLD are open to airborne surveys and underground mining (Zone 1).

Source: SRK, 2017

Figure 4-2: Location of the Pit Outline and Biological Reserve

4.2.2 Mineral Rights and Properties

Mineral exploration and mining are subject to the provisions of the *code minier*, which specifies that the State can grant to an operator a right to prospect or exploit the Mineral Resources over a specified area and period.

Special regulations have been established for French Guiana to take into account certain distinctions specific to this territory (law no98-297 of April 21, 1998). In addition to the *code minier*, that include Exclusive Research Permits (PER) for prospecting and Concessions for mining, the regulations concerning French Guiana provide for Mining Research Authorizations (ARM), in areas managed by the ONF, Exploitation Authorizations (AEX) and Exploitation Permits (PEX).

Mineral rights and mining are administered by the *Direction de l'environnement*, *de l'aménagement et du logement* (DEAL) under the authority of the Prefect. Their locations are reported in UTM, World Geodesic System RGFG95, Zone 22.

Exclusive Research Permit (PER)

In general, the PER is the initial permit application to conduct prospecting.

- Maximum area: No restriction. The area has to fit reasonably with the exploration objectives and the geological context.
- Dimensions & Form: No restrictions, as long as protected areas are not included within the area requested.
- Maximum period: 15 years. Initial application is for 5 years, twice renewable for up to 5 years. Surface area can be reduced by 50% in each renewal application. Following the extensions it is required to apply for a Concession or Exploitation Permit.

- Restriction: The initial application is open to competitor bidding if it covers an area greater than 50 km^2.
- Requirements: Financial commitments are based on the exploration program and expenditures proposed in the mining title application, which need to be in accordance with the surface area of the mining title. Conditions of renewal are based on the completion of the financial commitments in the corresponding period.

Exploitation Permits

Mining in French Guiana is permitted under the following permits:

- Concession;
- Exploitation Permit (PEX); and
- Exploitation Authorization (AEX).

PEX and AEX are exclusive to the *départements d'Outre-Mer* ("DOM"), (i.e. overseas deparments) such as French Guiana.

Concession

- Maximum area: No restriction.
- Dimensions & Form: No restrictions.
- Period: 50 years. Renewable for 25-year tranches if the mining operations are active at time of renewal. All of the concessions in French Guiana will expire by December 31, 2018. On the concessions, there are no financial commitments. However, for a concession to be eligible for renewal, its owner must prove actual gold production on the concession (by itself or by any company legally exploiting gold thereon) before December 31, 2018.
- Restriction: Open to competitor bidding unless it arises from a PEX or PER.

Exploitation Permit (PEX)

- Specific disposition: Medium-scale alluvial and small-scale vein-type mining.
- Maximum area: No restriction.
- Dimensions & Form: No restrictions.
- Maximum Period: 15 years. Initial application is for 5 years, renewable twice for up to 5 years per renewable term.
- Restriction: The initial application is open to competitor bidding unless it arises from a PER or if the total surface area is less of equal to 50 km^2 or less.

Exploitation Authorization (AEX)

- Specific disposition: Small-scale artisanal mining, mainly for alluvial exploitations, sometimes for primary gold in saprolite.
- Maximum area: 1 km^2.
- Dimensions & Form: 1 km x 1 km or 0.5 km x 2 km.
- Maximum Period: 8 years. Initial application is for 4 years, renewable only once for a term of up to 4 years.
- Restrictions: Maximum of 3 AEX by *département d'Outre-Mer* in a same 4-year period. An AEX can be issued over an area covered by a PER, Concession or PEX with consent of the

holder of these titles and as long as they are active. The holder of the PER, Concession or PEX loses all mineral rights over the area covered by the AEX.

The Project is composed of eight mining concessions and two PER, which cover an area of approximately 189.5 km^2 (18,949 ha). The two PER were granted on July 6, 2016, by decree of the French Minister of Economy, and published in the *Journal 44ctroy44r de la République française* on July 13, 2016 (JORF no0162). The concessions and PER are listed in Table 4-1 and shown in Figure 4-3.

Table 4-1: Land Tenure of Montagne d'Or

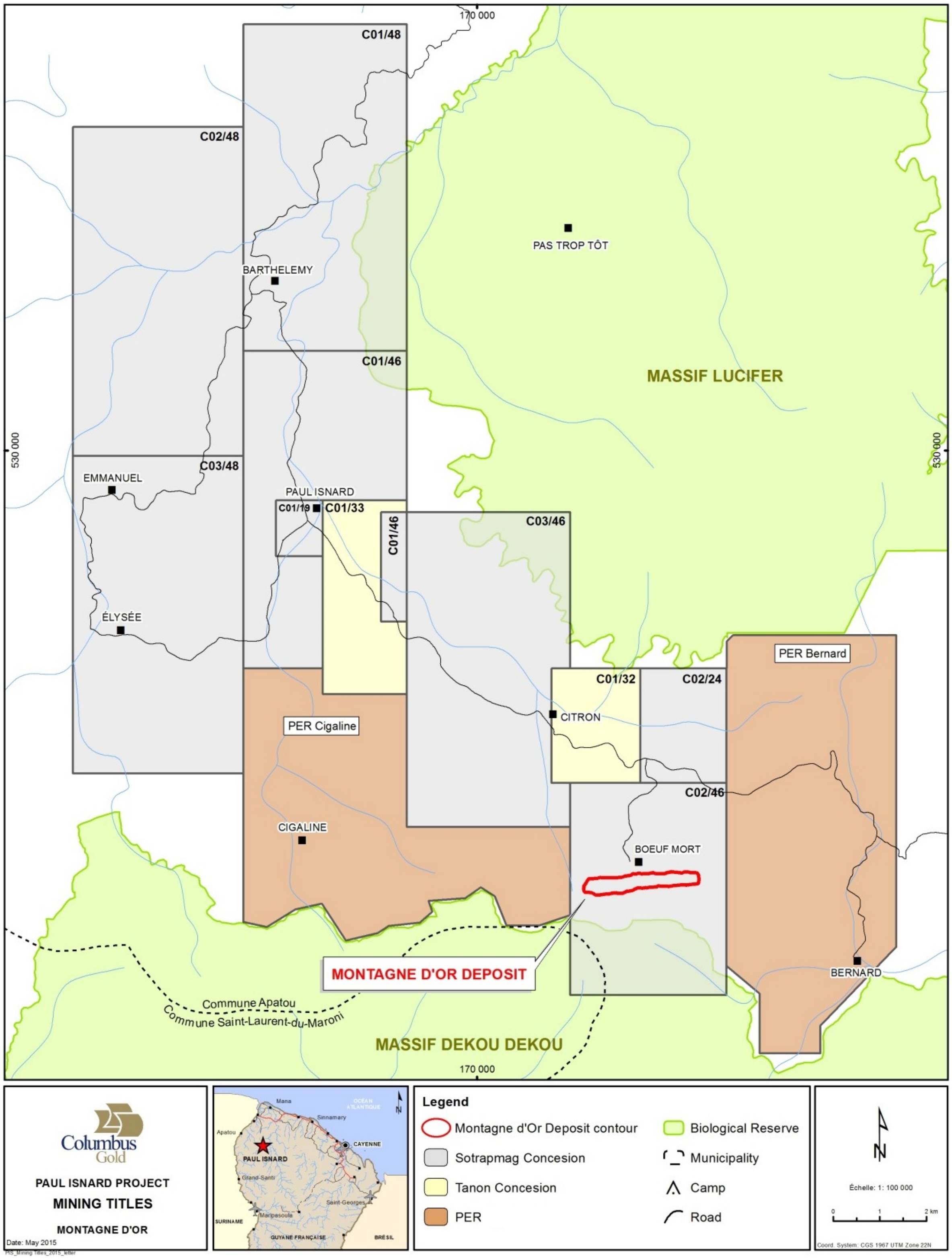
Permit Type	Company	Permit no.	Registration no.	Area* (km2)	Initial Grant	Transfert to Sotrapmag	Expiry date
Concession	Sotrapmag	692	C01/19	1.20	25/10/1919	27/12/1995	31/12/2018
Concession	Sotrapmag	25	C02/24	4.53	27/11/1924	27/12/1995	31/12/2018
Concession	Sotrapmag	214	C01/46	17.41	21/05/1946	27/12/1995	31/12/2018
Concession	Sotrapmag	215	C02/46	15.24	21/05/1946	27/12/1995	31/12/2018
Concession	Sotrapmag	216	C03/46	22.65	21/05/1946	27/12/1995	31/12/2018
Concession	Sotrapmag	217	C01/48	24.50	14/06/1948	27/12/1995	31/12/2018
Concession	Sotrapmag	218	C02/48	25.73	14/06/1948	27/12/1995	31/12/2018
Concession	Sotrapmag	219	C03/48	24.82	14/06/1948	27/12/1995	31/12/2018
			Total	**136.09**			
PER Bernard	Sotrapmag			29.00	13/07/2016		13/07/2021
PER Cigaline	Sotrapmag			24.40	13/07/2016		13/07/2021
			Total	**53.40**			
			TOTAL	**189.49**			

Source: Columbus, 2017

<u>Office National des Forêts Rights</u>

As most of the ground in French Guiana belongs to the French State and is covered by the equatorial rainforest, the Office National des Forêts (ONF) was designated to manage the private domain of the State. Therefore, any occupation of the ground, in forested areas, is submitted to an authorization by the ONF (camps, access roads, etc.). Subject to application, the ONF grants land use permits or "*Convention d'Occupation Temporaire du Domaine Privé de l'Etat pour activités minières*" (COTAM) to mining title holders. SOTRAPMAG holds a COTAM dated April 24, 2009, valid until December 31, 2018, for the use of the road from Apatou Crossing to Citron (60 km), for the surface area of Citron Camp and airstrip and all deforestation on the concessions. The COTAM has annual fees based on the surface area of the deforested land, kilometres of roads, and surface occupied. As an example, for the Project, SOTRAPMAG pays annual fees to the ONF for the use of the road from Apatou Crossing to Citron (€5,400), for the surface area of Citron Camp and airstrip (€3,700), as well as for the opening of new access roads and drill pads (variable, but about €800 for 2014). A *COTAM* will be necessary, in the future, for mine infrastructures and wastes and tailings sites.

Access to the Project mining concessions is guaranteed by the existence of the mining titles under the right of access to the Mineral Resource ("*accès à la 44iameter*").



Source: Columbus, 2015

Figure 4-3: Location of Columbus Concessions and PER

4.2.3 Nature and Extent of Issuer's Interest

In November 2010, Columbus entered into an option agreement to acquire control of the Project concessions from Auplata SA. In January 2013, Columbus completed the acquisition of a 100% interest in the Project.

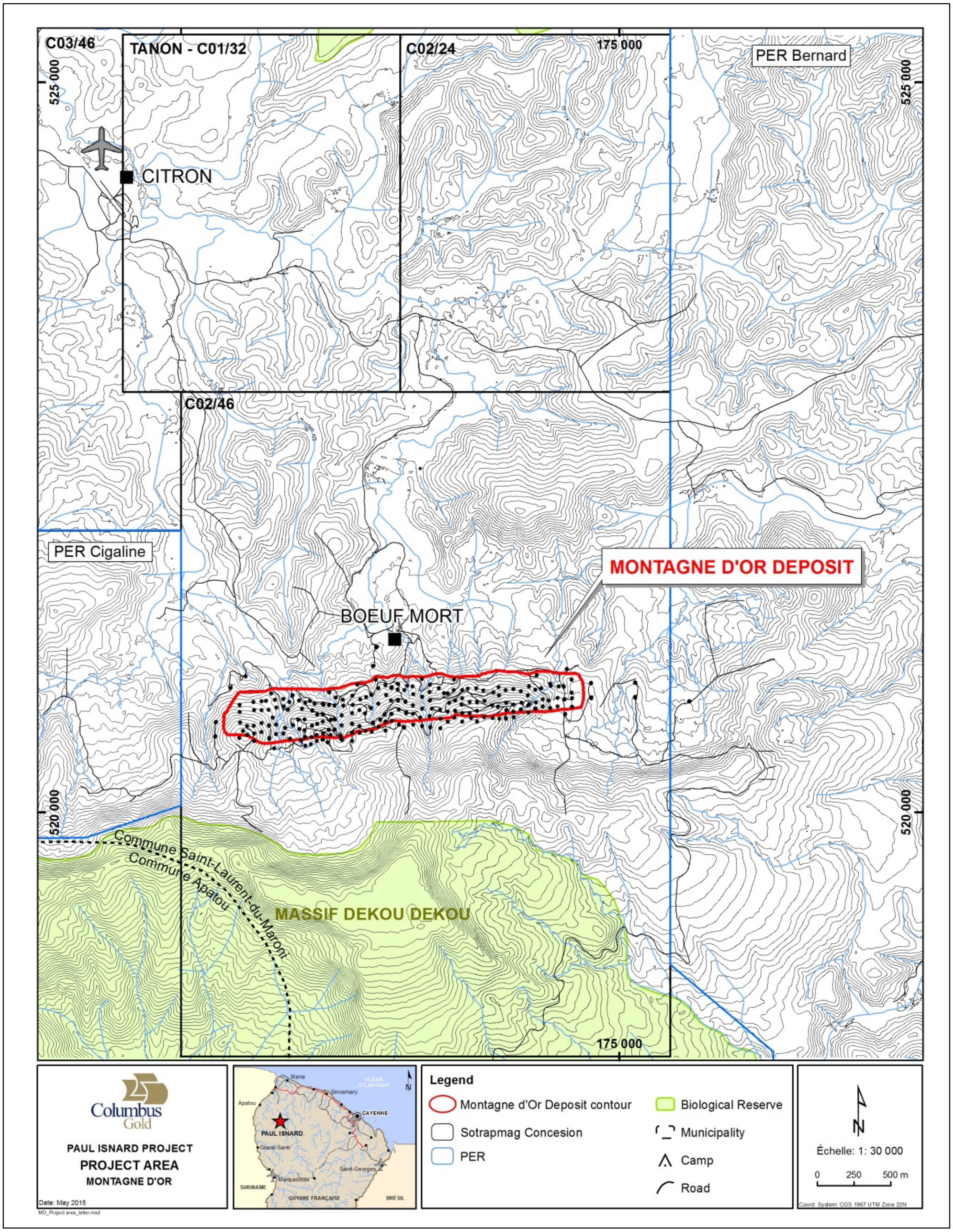
Columbus entered into a binding letter option agreement with Nordgold on September 17, 2013, and subsequently executed a definitive agreement on March 13, 2014, under which Nordgold has been granted the right to acquire a 50.01% interest in the eight Project mining concessions and the exploration permits. Nordgold can earn its interest in the mineral permits by completing a BFS and by expending not less than US$30 million in three years. During the earn-in period, up to January 14, 2016, Columbus was the Project operator and earned a 10% operator fee on certain expenditures. Effective January 15, 2016, Nordgold became the Project operator.

On January 12, 2016, Columbus entered into an agreement with Nordgold to sell a 5% minority interest in the Project for C$8,375,959 (US$6,000,000) (received). The formal acquisition and transfer of the 5% interest will not occur until Nordgold has funded completion of a BFS and achieved a minimum of C$32,730,000 (US$30,000,000) in Project expenditures, in order to earn an initial 50.01% interest in the Project. Nordgold will then deliver a notice of option exercise to Columbus to acquire its total 55.01% interest in the Project. If Nordgold does not earn its initial 50.01% interest, then Columbus is required to refund the advance of C$7,870,200 (US$6,000,000) and the sale of the 5% minority interest in the Project will not proceed.

4.2.4 Location of Mineralization and Facilities

The Montagne d'Or exploration area is located approximately halfway up the steep northern slope of the Dékou-Dékou Mountain within mineral concession C02/46 (215) shown in Figure 4-4. The mineralization and proposed mining and processing facilities, with the exception of the man camp, are within mineral concession C02/46.

The camp for the current exploration and the proposed mining operation could be located at Citron Camp. Citron Camp is within mineral concessions C01/46 held by SOTRAPMAG and C01/32 held by Tanon S.A. (Tanon). The access road crosses two Tanon held mineral concessions. The road crosses Tanon held mineral concessions C01/32 between the mineralized zone and Citron Camp and mineral concession C01/33 north of Citron Camp (Figure 4-4). Under the mining code, SOTRAPMAG has rights to any access roads leading to the Project concessions.



Source: Columbus, 2015

Figure 4-4: Montagne d'Or General Site Map

4.3 Royalties, Agreements and Encumbrances

The Project is subject to a 1.0% NSR royalty payable to Sandstorm Gold Ltd.

There is also a NSR royalty of 1.8% on the first 2 Moz of gold produced and 0.9% on the next 3 Moz of gold produced on the Project payable to Euro Ressources, an 86%-owned indirect subsidiary of IAMGOLD Corporation.

The state royalty on gold production payable in French Guiana are set on a yearly basis by articles 1519, 1587 and 1599 of the General Tax Code. There is a Municipal tax of €137.90/kg; a Departmental tax of €27.50/kg; and a French Guiana royalty of €672.01/kg, respectively.

The Project is also subject to reclamation of previous mining works, as described in Section 4.4.1, to a maximum expenditure of €350,000. The reclamation work started in October 2014 and completed on February 29, 2016.

4.4 Environmental Liabilities and Permitting

4.4.1 Environmental Liabilities

The Project area is an intermittently active exploration property centred in dense tropical rain forest. Exploration activities require access road and drill pad construction, trenching, water management features, as well as construction of worker camps. Environmental liabilities resulting from previous and ongoing exploration activities are fairly limited due to the high precipitation and rapid natural rehabilitation that occurs in the rainforest. Holders of exploration permits are required by law to reclaim worked areas, control stormwater and potential sedimentation of downstream surface water resources, and are strictly prohibited from using mercury. These conditions are monitored closely by the government.

Illegal artisanal placer mining that occurs over much of the Project area has disturbed considerable land area, and continues to impact local surface water resources through increased sedimentation and mercury contamination.

4.4.2 Required Permits and Status

Discussion related to mining in French Guiana, the Mining and Environmental Codes, as well as the permits and authorizations necessary for mineral exploration and exploitation is provided in Section 20.7. In addition, some background into the anticipated mining code reforms is also provided.

4.5 Other Significant Factors and Risks

There are no known factors or risks that affect access, title or right or ability to perform work on the property.

5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

5.1 Topography, Elevation and Vegetation

Large areas of the general site topography are relatively flat lying, but there are some higher elevation areas with steeper slopes, as described in the Physiography Section 5.5. The mean elevation of the Project site area is approximately 130 metres above sea level (masl).

Most of the region is covered by a thick canopy of primary and secondary tropical forest. The larger valleys have been worked by alluvial miners in the past and are generally covered by thinner secondary forest or grassy-scrub and bamboo. Thick areas of bamboo are also present in many streams especially on the steeper slopes and in areas of old mine workings.

5.2 Accessibility and Transportation to the Property

The population of French Guiana in 2013 was 244,118. The nearest substantial population centre to the Project is SLM. Official figures from the 2013 census give the town a population of 41,515.

Project materials and procured equipment will be delivered to the Project site via the ports of Cayenne or SLM. Both port facilities have limited materials handling capacity and no heavy lift capacity. Accordingly, it will be necessary to deliver the materials and equipment in self-unloading vessels and unload directly on to road transport located alongside the vessel. Open laydown areas are available close to the Cayenne port.

Alternatively, the port of Paramaribo in neighbouring Suriname could be used but this would require a ferry crossing at SLM. It may also require a transfer from Suriname based trucks to those belonging to local transport providers.

Barges are available in the region and trans-shipment from Paramaribo and Cayenne by barge to SLM is feasible.

Based upon an assessment of road conditions between Cayenne and SLM, the port of Cayenne will be restricted to receiving standard gauge materials (i.e. 6.0 m and 12.0 m containers) with a payload maximum of 28 tonne. The existing transport corridor from Cayenne to SLM has a number of weight restricted bridges and within a section of the road between Laussat and SLM there are dimensional restrictions as a consequence of a truss bridge.

The Project is accessible via a 120 km seasonal forest road from the town of SLM, where the port of St. Laurent is located, or by helicopter/light aircraft to the Project's base camp at Camp Citron.

The current condition of the public section of the road between SLM and Apatou Crossing road is fair to poor and will need repair and maintenance during the Project construction and on-going operation phase. The general road condition and increase in construction/mining traffic may accelerate deterioration of the road and give rise to a heightened potential of incidents with local traffic. An upgrade of the access road from SLM to Apatou Crossing is to be considered.

The access road from Apatou Crossing to Citron is a private road although it is used by multiple organisations. It is in very poor condition and impassable to vehicles without 4WD and even light 4WD vehicles have difficulty negotiating the road during the worst of the wet season. The scope of works includes the upgrade of the existing main access road from Apatou Crossing to Citron, a distance of approximately 54 km.

5.3 Climate and Length of Operating Season

The climate is equatorial, with daytime temperatures between 29°C and 33°C, decreasing to 19°C to 23°C at night. There are two wet seasons; the main period is typically from April through July, and the lesser period lasts from December to mid-March. Average annual rainfall is in excess of 2,000 mm with a minimum monthly rainfall of 50 mm. Humidity is constantly high and typically ranges between 78% and 92%.

The Project area experiences lengthy periods of high rainfall amounts and precipitation, and run-on inflows are expected to be significant. The operating season is year-round.

5.4 Resources and Infrastructure

Existing infrastructure at site while suitable to support a limited exploration and drilling program is inadequate to support construction or ongoing operation of a project of this size.

While several large international contractors have regional offices in French Guiana local resources in the way of skilled labour and construction equipment are limited. Having noted that, the Guiana Space Centre, a French and European spaceport has been operational since 1968 and has been progressively expanded and developed to support both the European and Soyouz space programs.

Ongoing operations will be supported out of the town of SLM, the main regional population and administrative centre as well as the location of French Guianas second port. SLM has a population of over 40,000.

In the short term, fuel deliveries will be out of Cayenne but a subsidiary depot close to SLM is planned for construction by about 2020.

None of the current logistical or regional capacity/capability limitations are incapable of being overcome and their limitations have been factored into the development of the study.

5.5 Physiography

The general relief of the region is dominated by three geomorphological features:

- The east – west trending Massif Dékou-Dékou Range (south of the Project);
- The southwest – northeast trending duricrust plateau of Montagne Lucifer, average elevation 563 masl (north of the Project site); and
- The northwest – southeast drainage system of the Roche River.

The Project site occupies the northern flank of the Dékou-Dékou Range.

There are numerous broad valleys, many of which have been exploited for their alluvial gold deposits. These are separated by areas of moderately rugged to more rounded hilly relief and often deeply incised valleys.

5.6 Sufficiency of Surface Rights

There are sufficient surface rights for access and construction of the Project.

The Tanon concession (Number C01/32, located at the TSF and near to the camp, Figure 1-1) is set to expire on December 31, 2018 and no application to renew it has been received by the French authorities at the required deadline (December 31, 2016). Nordgold should be able to straightforwardly acquire the grounds.

As per the Mining Code, a mining concession provides exclusive surface and subsurface rights to a mineral deposit, while it is not required to hold a mining concession for the purpose of mining infrastructures, including a processing plant and storage facilities.

As per Environmental Code, the ICPE permit to operate (Installation Classified for the Protection of the Environment) requires that the zoning of the Local Urbanism Plan (PLU) is suitable for the mining activities and that the land owner, the ONF, on behalf of the French State, gives consent for the occupation of the land for these activities by way of an occupancy agreement of the State Forest Domain (COTAM).

5.7 Availability of Power, Water and Services

The French Guiana grid is not connected to the grids of any neighbouring countries. In 2003 electricity production was 465.5 GWh from a mix of hydroelectric and fossil fuel generation sources.

A 106 km, 90 kV overhead powerline will connect the site to the existing grid. The point of connection is at the Margot HV substation located outside SLM. It is anticipated that by the time the Project is constructed a new regional power station will be built in the vicinity of SLM to supplement the current mix of hydro and fossil fuel generated power.

With high rainfall experienced in the area, the Project will source all of its water requirements from the open pit and run-off from disturbed areas. By the time the Project commences operation it is anticipated that run-off will exceed water requirements and a treatment plant will be installed to treat and release surplus run-off (contact water) to local waterways.

The Project will have to be self-sufficient in providing all services including fuel supplies, accommodation, communications, potable water, sewage and waste disposal.

French Guiana has limited capacity to support either construction or operation of a project of this size. Construction equipment and materials will have to be imported and the Project will have to be self-sufficient in workshops, stores and maintenance equipment requirements.

Several large international contractors have regional offices in French Guiana, but local resources in the way of skilled labour and construction equipment are limited. Site operations will be supported out of the town of SLM.

5.7.1 Potential Tailings Storage Area

SRK evaluated several TSF areas in the Project area for storage capacity and embankment volumes. SRK selected a site north of the planned open pit and WRDs, which is centrally located in a broad valley setting, and relatively close to the processing plant facilities.

5.7.2 Potential Waste Disposal Areas

SRK selected two main sites for WRD locations, both immediately north of the planned open pit. The WWRD will be constructed over two phases, and the CWRD will be constructed over of four phases. An access corridor will run between the two planned WRDs.

Locations are also planned for a low-grade ore stockpile and topsoil stockpiles.

5.7.3 Potential Processing Plant Sites

The plant site location selected for the earlier PEA was re-evaluated early in this study phase, the original site having a number of shortcomings namely its proximity to the eastern corridor between the Dékou-Dékou and Lucifer biological reserves, the requirement for roads and powerlines to cross the WRD and TSF sites from west to east, and the distance from the western end of the open pit which is seen as the end most likely to host any future extensions of mineralization.

An alternative site was therefore selected to the north-west of the pit and to west of the WWRD on the side of a north-south valley that will become the main route for road access between the accommodation camp and the powerline.

The plant site has been located on the edge of and above the floor of the valley. In this location stormwater run-off will be directed away from the plant and into the valley watercourse.

To the west and slightly below the process plant site is an area set aside for the main HV switchyard and the WTPs. To the north is the MSA.

The selected site is close to the open pit and low grade stockpiles to minimize haul distances and provides a good balance of being remote enough from the accommodation camp to avoid unnecessary noise or other disturbance but is close enough that the daily 'commute' to work will be minimal.

6 History

Section 6 has been excerpted from the Coffey 2014 Technical Report. Standardizations have been made to suit the format of this report. Changes made by SRK are indicated by the use of brackets [ ] or in sentences containing "SRK". Some spelling has been modified.

6.1 Prior Ownership and Exploration

The Paul Isnard concessions have been a regional centre of alluvial and colluvial gold production since 1873 with some minor underground development in a few places. Beginning about 1890 bucket type dredging was undertaken and was replaced by dragline operations in 1949. Due to government permitting issues, little if any work was undertaken except by small illegal miners from 1950 to 1965 when placer mining recommenced and continued until approximately 1997.

The area was previously explored by the Bureau Minier Guyanais (BMG) and later the Bureau de Recherches Géologiques et Minières (BRGM), the French Geological Survey. This work confirmed the alluvial mining potential of the region and also located the primary Montagne d'Or prospect as a result of a regional geochemical program in 1976. This was not recognized as such until the data was reinterpreted in 1984. The BRGM undertook detailed surficial geochemical work and geological mapping.

The Paul Isnard Mine was started in 1956 by a company called SERMIG; gravel mining commenced in 1966 and continued for 20 years through an American company. Recovery was through an amalgamation plant and must have been poor. From 1986, a new owner (Pichet-Driss) obtained control, improved the process and operated the mine until 1993. SOTRAPMAG was involved in the gravel mining operation as a partner with the SGM, CERMI and Pichet-Driss.

In May 1993 Golden Star Resources Ltd endeavored to acquire title to the mine properties of the Paul Isnard Mine off SOTRAPMAG who was the owner of the mine and carried out a two-week evaluation of the operation. Total production from 1987 to 1993 was at this stage reported at 5,142 oz of gold and 354 oz of silver. This would roughly indicate a 7% average silver content of the gold doré.

Intensive exploration did not begin until 1994 when Guyanor Ressources S.A. ("Guyanor" approximately 70% owned by Golden Star Resources) had acquired the concessions and undertook regional scale remote sensing (LandSat, geophysics), geological examinations and geochemical surveys. Guyanor acquired the property in October 1994 through the 100% acquisition of the mining company SOTRAPMAG. Guyanor is registered in French Guiana with the right to explore deposits of gold, precious metals, base metals, and precious stones.

When Guyanor purchased SOTRAPMAG, it paid off an interest of Alcatel Alsthom Compagnie Générale d'Electricité (ALCATEL) in a primary deposit in the area to the BRGM while the company La Source Développement (LaSource) received an initial 25% participating interest. It is reported that LaSource did decide not to participate as a minority partner and that its interest was subsequently diluted.

From June 1996 until May 1998 exploration on the property was operated as a joint venture between SOTRAPMAG and Asarco Guyane Française with LaSource as a non-contributing partner. A PER was granted by Ministerial Decree (Official Bull. Dated November 30, 1999) 100% to Guyanor (later named Euro Ressources) on November 26, 1999 for a period of three years from 1 December 1999 to 30 November 2002. Following the formation of the Joint Venture with Asarco and La Source, detailed geology, geochemistry and geophysics was completed along with 56 drillholes totaling for 10,916 m. In September 1999 the LaSource interest is reported as approximately 10% and that when it falls to below 10% it will convert to a 2.5% net proceeds royalty.

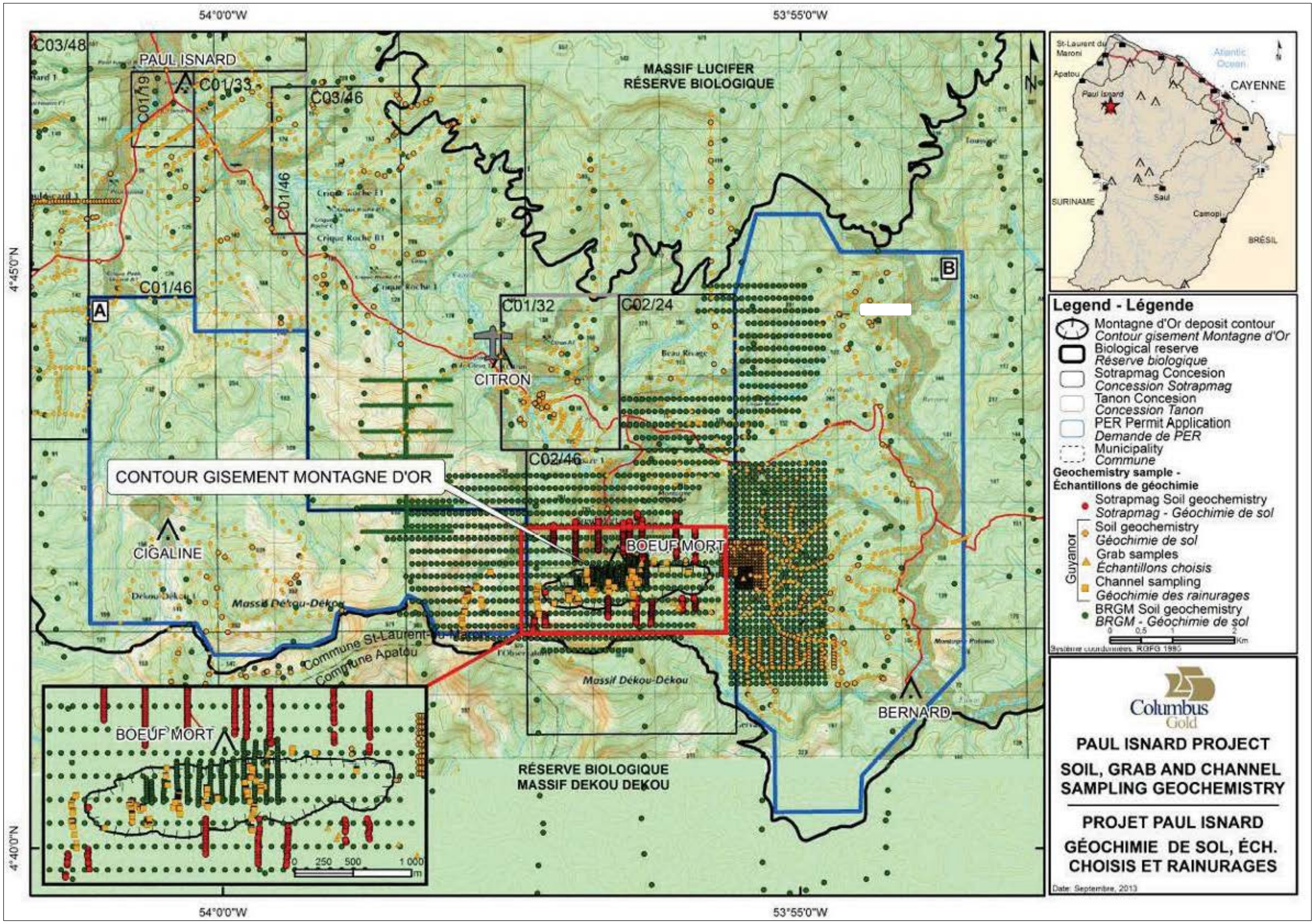
In 2001 a program of drilling was completed by Guyanor in conjunction with a JV agreement signed between Guyanor and Rio Tinto Mining and Exploration Ltd. Rio Tinto however concluded that the deposit did not have sufficient potential (more than 10 Moz) within saprolitic and near surface material to be mined by open pit methods followed by a cyanide recovery process.

Input to this study was mainly a re-interpretation of all available structural, geological and geophysical data and a study of older drill core obtained by Élysée (six diamond drillholes for 598.45 m) and Apollon (three diamond drillholes for 405.40 m), a regional geochemical soil program covering areas that were not previously covered (total of 1,058 soil samples) and a follow-up soil geochemistry and ground geophysics program (69 km) investigating the located anomalies. Selected anomalies were followed-up in 2001 with a limited diamond drilling program (Élysée six additional drillholes for 636.50 m, Paul Isnard three drillholes for 358.95 m, Citron three diamond drillholes for 343.50 m). One drillhole at Paul Isnard (Montagne d'Or) intersected a 7.0 m mineralized interval at 1.03 g/t Au. After completion of the program, Rio Tinto decided to withdraw from the JV.

Guyanor has carried out exploration activities in the areas at and around Montagne d'Or since 1994. Diamond drilling by Guyanor from 1996 (in JV with Asarco) to 1998 resulted in a total of 56 drillholes for 10,916 m. Guyanor also drilled 18 holes in 2001 in a JV with Rio Tinto and in 2007 Euro Ressources drilled one additional drillhole at Paul Isnard. Guyanor became Euro Ressources.

Until the property was acquired by Columbus in 2010, work done largely consisted of desktop evaluation of the resource potential and possible economic viability, and little additional exploration work was undertaken.

Since before 1900 up to around 1950, small scale alluvial mining has taken place in the area. This was followed by large scale alluvial mining from 1965 while the BRGM undertook geological mapping and regional geochemistry from 1930 to around 2000. Guyanor started work on the property in 1994. A regional overview of the various soil sampling, grab sampling and channel sampling programs is provided by the map in Figure 6-1.



Source: Coffey, 2014

Figure 6-1: Plan Map Overview of Historic Exploration Campaigns

6.2 Historical Mineral Resource Estimations

There have been six previous Mineral Resource estimations, prepared in accordance with CIM, for the Montagne d'Or prospectMineral Resource. These are summarized in Table 6-1. SRK notes the historical resources are not current Mineral Resources; they have been superseded by the current SRK Mineral Resource estimate discussed in Section 14 of this Technical Report. SRK has not done sufficient work to classify the historic estimates as current Mineral Resources. Columbus and Nordgold are not treating the historical estimates as current Mineral Resources. The historical resources are provided here for information purposes only.

Table 6-1: Previous Resource Estimates for the Montagne d'Or Deposit

Year	Source	CIM Compliant	Resource Classification	Cut-off (g/t)	Tonnes (M)	Au (g/t)	Contained Au oz (M)
January 2004	RSG Global	Yes	Inferred	0.8	60.5	1.5	2.9
February 29, 2008	SRK	Yes	Inferred	0.5	33.2	1.7	2.0
February 11, 2011	SRK	Yes	Inferred	0.4	36.7	1.6	1.9
November 23, 2012	Coffey Mining (Canada)	Yes	Inferred	0.3	115.2	1.44	5.3
August 4, 2014	Coffey Mining (Australia)	Yes	Inferred	0.3	169.2	0.9	4.6
June 2, 2015	SRK	Yes	Indicated	0.4	83.2	1.45	3.9
July 8, 2015	SRK	Yes	Inferred	0.4	22.4	1.55	1.1

Source: SRK, 2017

7 Geological Setting and Mineralization

Section 7 has been partially excerpted from the Coffey 2014 Technical Report and Updated by Columbus current to this report. Standardizations have been made to suit the format of this report.

The Montagne d'Or deposit is composed of a bimodal felsic and mafic igneous units with lesser volcaniclastics towards the base of the sequence. The units strike east-northeast and dip steeply south. The eastern portion contains a preponderance of mafic volcanics relative to felsic volcanics. All geological units have been strongly deformed, as evidenced by a penetrative S1 foliation that locally transposes S0 and in places is mylonitic. The volcanic-plutonic package that hosts the deposit is tightly to isoclinally folded. The S1 foliation is constant throughout the section, striking on average 084° with an average 72°S dip. The intensity of deformation varies significantly over the distance of a few metres. The Project area is cross cut by post deformation diabase dikes that were apparently emplaced within northeast striking shears, faults or fractures that formed during a regional transcurrent tectonic event.

In general, the Montagne d'Or deposit consists of a number of tabular mineralized bodies within laminated, mainly felsic metavolcanic rocks. Mineralization has been encountered over a strike length of almost 2,500 m and to a vertical depth of at least 200 m. The mineralization is open at depth, along strike and internally between widely spaced holes.

The mineralization appears as narrow elongated higher grade lenses within broader zones of low grade but anomalous mineralization (0.25 to 0.4 g/t Au). The main area of gold mineralization occurs in a series of generally east-northeast striking parallel zones with overall dimensions of 2,200 m x 400 m wide and to at least 200 m vertical depth. However, gold has been encountered outside the main zone of mineralization in the host rocks over a strike length of at least 3,500 m. Several distinct anomalous mineralized domains can be recognized that are separated by barren intercalated mafic and felsic rocks. Mineralization consists of semi-massive sulphide bands, as sulphidic stringers and as disseminated sulphides. Visible gold is present but rarely observed; preliminary mineralogical work suggests that it occurs along micro-fractures and on sulphide grain boundaries.

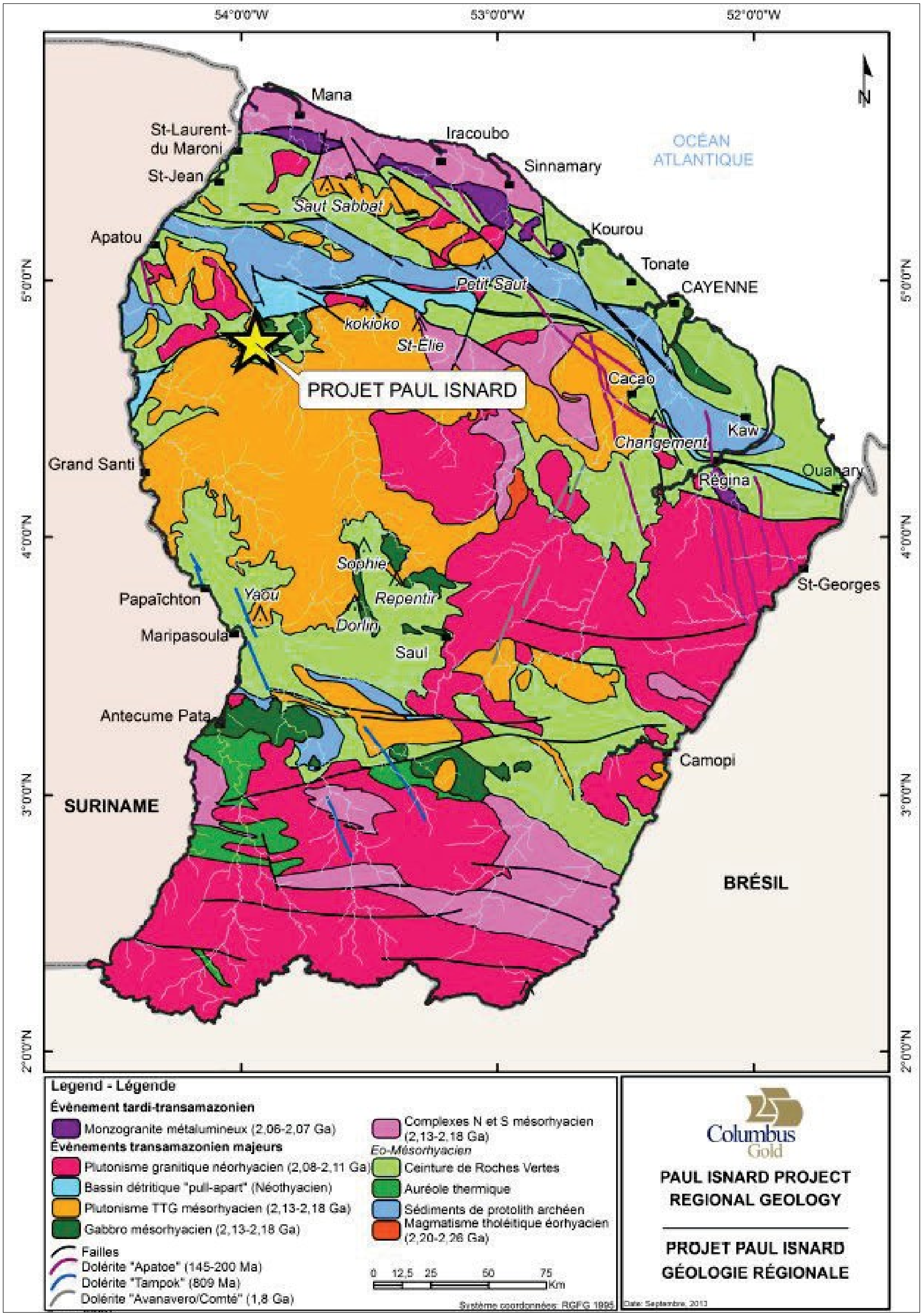
7.1 Regional Geology

The following is based mainly on work published by Milesi et al (2003) and Delors et al (2001), and on the most recent geological and structural interpretations carried out by a team from the Université du Québec à Montréal (UQÀM) and published in 2014 (Giraud et al, 2014). The latter studies also use and discuss historic and important geological interpretations by Vanderhaeghe et al (1998), and Franklin et al (2001). An earlier publication important for understanding the evolution of the geological interpretation of the French Guiana geology is the exploration report by Suter prepared for Guyanor in 1999.

The Project concessions occur within the Guiana Shield, a large (approximately 900,000 km^2) segment of the Amazonian Craton of South America (Figure 7-1). The majority of the Guiana Shield formed during Proterozoic periods of intense magmatism, metamorphism and deformation that culminated in the Transamazonian tectono-thermal event of 2.1 to 1.9 Ga. The low-grade, volcanic-sedimentary greenstone sequences and affiliated granite intrusives that comprise the shield yield U-Pb age dates between 2.25 Giga-annum (Ga) and 2.08 Ga.

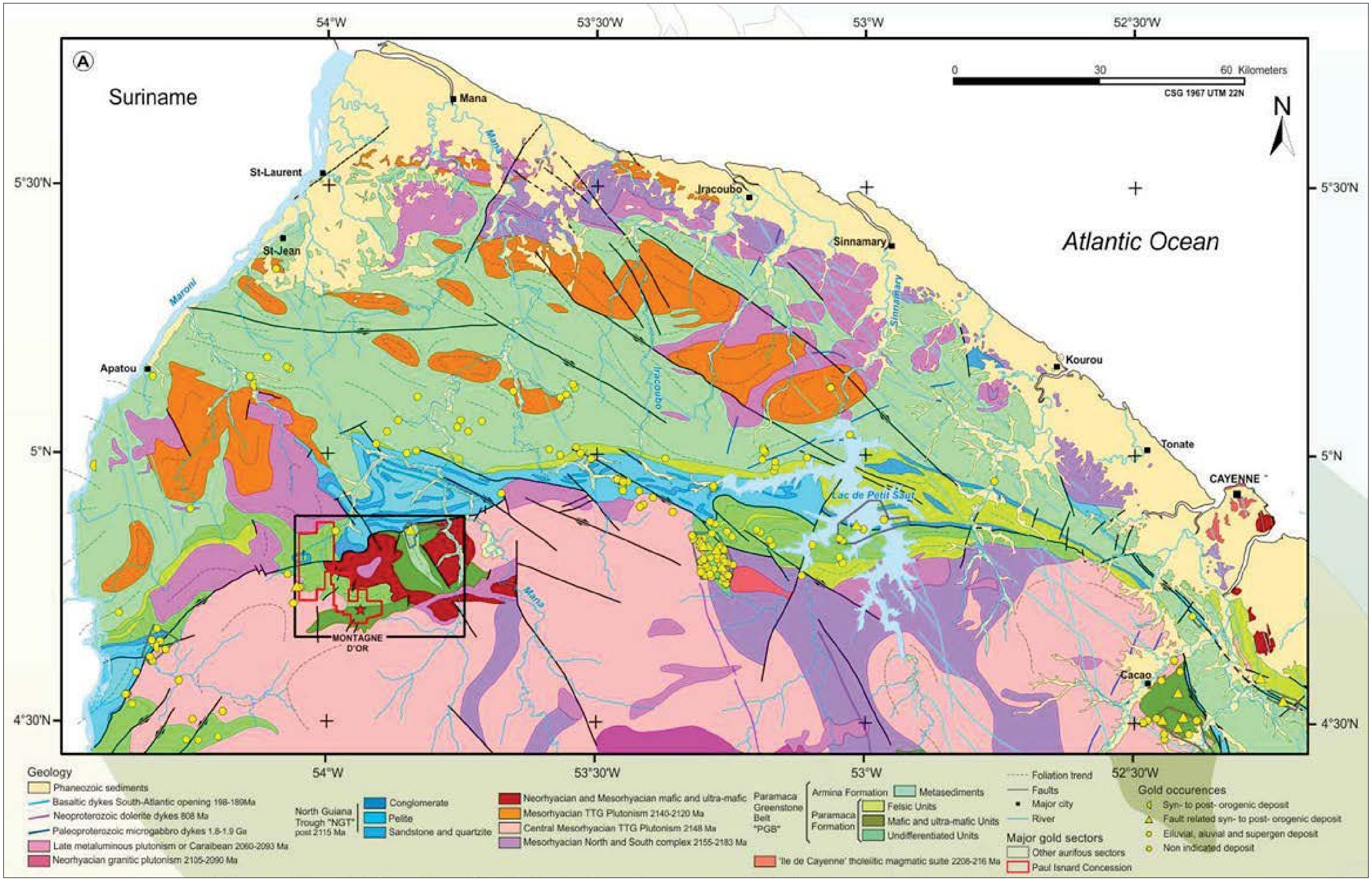
Major structural features include the Central Guiana Shear Zone (CGSZ) and the North Guiana Trough (Sillon Nord Guyanais, NGT). The CGSZ is a large-scale ductile shear zone, extending from French Guiana westerly through central Suriname and north-central Guyana. The NGT is interpreted to be a sinistral strike-slip "pull-apart basin" formed during one of the major tectonic stages of the Transamazonian Orogeny (Voicu et al, 2001).

The greenstone belts of French Guiana are divided into two major groups. The northern group is associated with the NGT and includes the Lower Proterozoic Paramaca Greenstone Belt (PGB), a formation consisting of volcanic, volcaniclastic and sedimentary units. The PGB trends roughly from the west to the east through British Guiana, Dutch Guiana (Surinam) and French Guiana (Figure 7-2).



Source: Coffey, 2014

Figure 7-1: Large Scale Geological Map of French Guiana



Source: Coffey, 2014

Figure 7-2: Large Scale Overview of the Geology of Northern French Guiana, showing the location of the Montagne d'Or Project

Together with intrusive complexes of tonalite, trondhjemite and granodiorite, the PGB forms the Guiana Shield which was connected during the Paleozoic to the West African Shield (after Guiraud, Jébrak and Tremblay, UQÀM, April 2014). The PGB is interpreted as the remnant of a volcanic island-arc sequence that was tectonically deformed during the Transamazonian Orogeny, interpreted to be the result of plate convergence between the West African and the Guiana Shields.

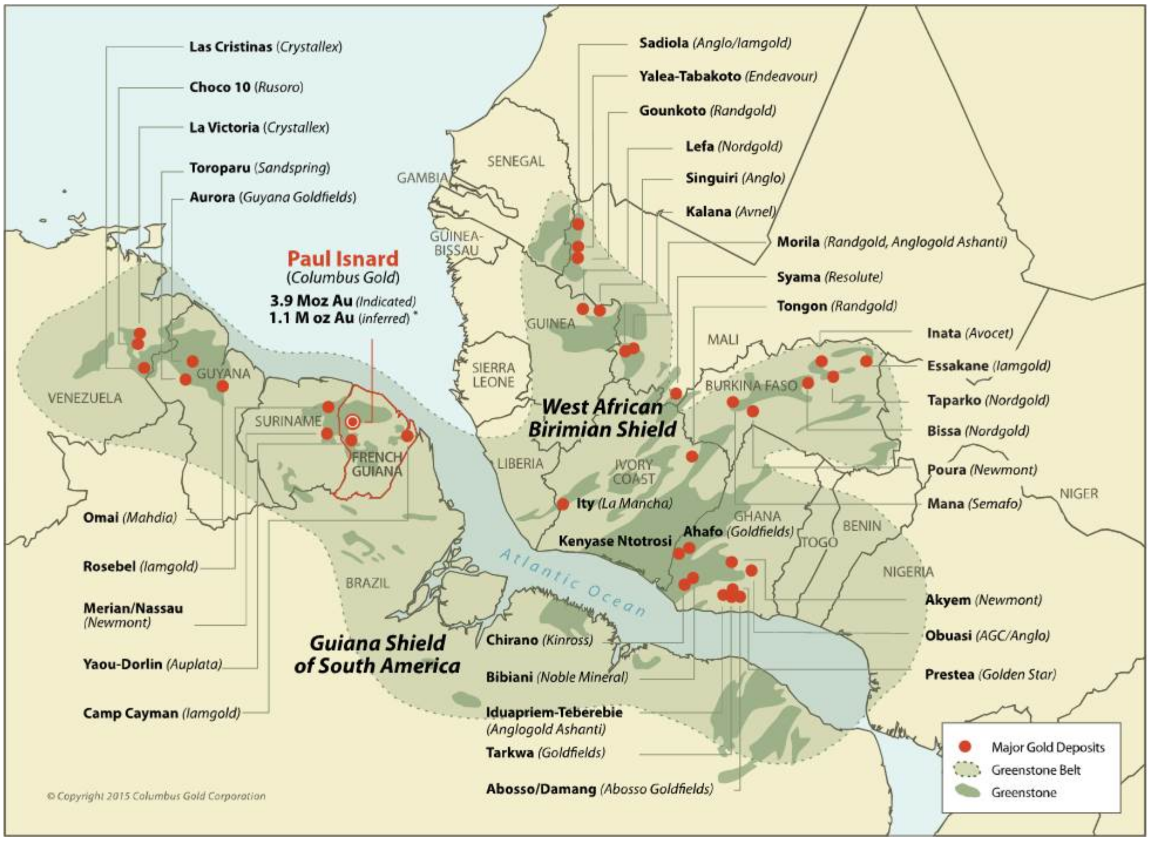
This PGB occurs extensively across northern French Guiana, striking N110°E and hosting a number of gold deposits including Montagne d'Or, Camp Caiman, St. Elie, Koolhoven and Rosebel in Surinam. The southern group is associated with the CGSZ and extends from Surinam through French Guiana. It includes sedimentary rocks of the Lower Orapu Formation and volcanic-sedimentary units of the Arima Formation (2.11 to 2.09 Ga), which unconformably overlie volcanic units of the PGB and the granite-gneiss complex of the Guianese Massif Central (2.3 to 2.2 Ga and 2.13 to 2.08 Ga). This southern group hosts gold mineralization at Benzdorp in Surinam, Yaou and Dorlin in French Guiana, and numerous other smaller workings. Most of the remainder of French Guiana is composed of the Lower Proterozoic granite-gneiss metamorphic complex of the Guianese Massif Central, and a central belt of Paramaca volcanic, volcaniclastic, and sedimentary lithologies.

The northern and southern domains of the PGB are separated by the intrusions of tonalite, trondhjemite and granodiorite (TTG). Along its northern boundary, at a distance of approximately 15 km from Montagne d'Or, the PGB is bounded by sandstones and conglomerates of the NGT. Along the southern margin, the Greenstone Belt is in contact with large intrusive domes of TTG.

The PGB is locally limited to the south and west by regionally extensive post-orogenic granites and to the east by Inferred high-grade metamorphic rocks of migmatitic and granitic gneiss. To the north, a narrow band of Paramaca-Armina Formation is unconformably overlain by the Upper Detrital Series (Ensemble Detrique Superieur EDS), silici-clastic sediments comprised of the Bonidoro, Orapu and Rosebel Formations. The EDS are surrounded by gabbro and granite and are interpreted as having been deposited in pull-apart basins associated with the NGT.

The felsic-mafic metavolcanic rocks of the PGB are overlain by the Armina Formation, a series of alternating sedimentary rocks (sandstones, graywackes and pelites); however, this formation has not been intersected by drilling in the Project area. The BRGM obtained a radiometric age in the Project area of 2,152 ± 8 Mega-annum (Ma) from a rhyolite which provides a possible date for the volcanic series however the age of the mineralization is unknown. Locally, gabbro intrusions occur which have yielded radiometric dates of 2,150 Ma to 2,145 Ma, similar to the TTG.

The PGB and EDS are probable equivalents or correlatives of respectively the Birimian and Tarkwaian sedimentary sequences of the West African Shield and may have been co-extensive prior to the separation of Gondwanaland in the Mesozoic (Figure 7-3). The Project lies within the northern PGB and is comprised of mafic and felsic metavolcanic rocks of the Paramaca Formation.



Source: Columbus, 2015

Figure 7-3: Map Showing Correlation of the Guiana Shield with the West African Birimian Shield

7.2 Property Geology

7.2.1 General

Montagne d'Or occurs within a bimodal felsic-mafic series of Proterozoic volcanic rocks, cut by slightly younger felsic to intermediate composition intrusive rocks. The gold mineralization is hosted within a 400 m thick, tightly to isoclinally folded sequence of predominantly felsic and lesser mafic volcanic rocks. The units strike east-northeast, dip steeply south and are exposed on the northern slopes of Dékou-Dékou Mountain.

The eastern portion contains dominantly mafic volcanics with less felsic volcanics than are present in the western parts. The mineralized units have been strongly deformed, as evidenced by a penetrative S1 foliation that locally transposes S0 bedding and in places is mylonitic. The orientation of the S1 foliation is constant throughout the section, striking on average 084° with an average 72°S dip. The intensity of deformation varies significantly over the distance of a few metres. The deposit is cross cut by post mineralization/deformation diabase dikes.

The volcanic complex of Montagne d'Or is bounded in the north by mafic volcanics, granite and gneiss and is bounded along its southern margin by mafic volcanics and metasediments that were thrust over the volcanic package. Several slivers of detrital metasedimentary rocks are locally faulted or folded within the overthrust mafic volcanics. The metavolcanic rocks have metamorphosed to greenschist grade.

The entire region has undergone Tertiary age lateritic weathering which resulted in a saprolite cover of varying thickness and in which variable downslope movements have taken place.

7.2.2 Lithology

The Montagne d'Or deposit is hosted within a tightly to isoclinally folded, steeply south dipping lithological package consisting of felsic and mafic metavolcanic rocks that are assigned to the PGB. The mafic metavolcanic rocks were previously divided into two units, a Lower Mafic Unit that lay to the north of the deposit and an Upper Mafic Unit that comprised the eastern part of the deposit (Coffey, 2014). Here, a single mafic metavolcanic unit is interpreted (Figure 7-4). The grouping of both of the previously defined mafic units into a single unit is justified by the paucity of data that are available for the region to the north of the deposit. The metavolcanic package is intruded by three distinct felsic to intermediate plutonic units that host minor amounts of gold; from oldest to youngest these are granodiorite, quartz-feldspar porphyry and feldspar porphyry. Quartz-carbonate veins occur throughout the deposit but do not contain significant mineralization.

To the south, the Montagne d'Or deposit is structurally overlain by a tightly folded and metamorphosed volcano-sedimentary package (TGC 2016b). The metasedimentary rocks consist of graphitic argillites as well as siltstones and fine grained greywackes. The interstratified mafic metavolcanic rocks include massive mafic flows and mafic tuffs. On its northern side, the Montagne d'Or deposit is structurally underlain by a mafic volcanic – sedimentary package similar to the one that lies to the south. The metavolcanic and metasedimentary units underwent greenschist grade peak metamorphic conditions. Whole-rock geochemistry data show that the felsic lithologies have a calc-alkaline chemistry and were likely deposited in an arc or back-arc basin environment. Whole rock compositions range between granite and granodiorite (Suter, 1999; GoldFields, 2001).

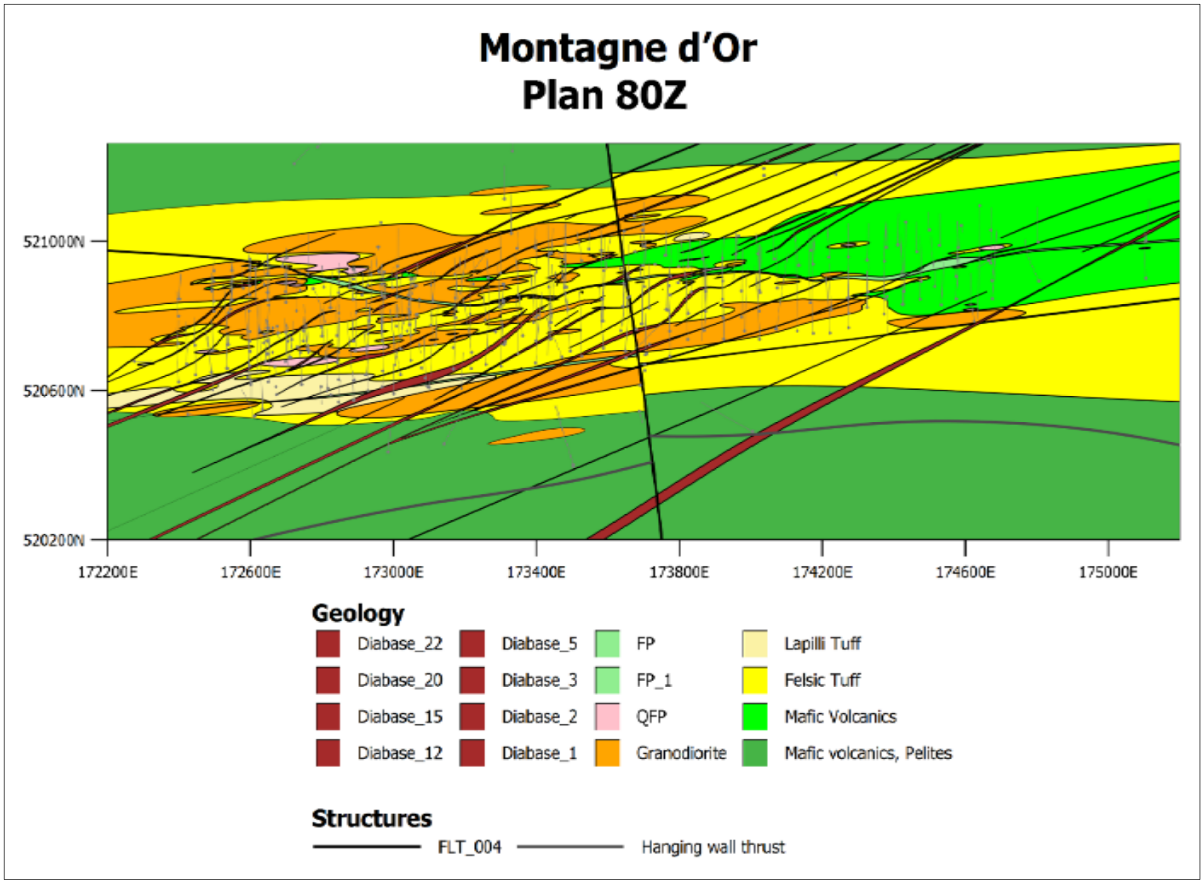
All units described above are cross-cut by a series of northeast striking diabase dikes.

Nearly 50% of the mineralization at the Montagne d'Or deposit is hosted by felsic metavolcanic units, mainly the Felsic tuff unit as defined here.

The tightly folded metavolcanic and plutonic rocks that represent the geology of the deposit can be assigned to the following principal units, listed from oldest to youngest, and that are described in the following paragraphs:

- Mafic metavolcanics;
- Felsic tuff;
- Lapilli tuff;
- Granodiorite;
- Quartz-feldspar porphyry; and
- Feldspar porphyry.

A schematic of the local geology of the Montagne d'Or prospect is shown in Figure 7-4.



Source: Columbus, 2015

Figure 7-4: Schematic Overview of Main Local Geological Units

Mafic Metavolcanics

This unit occurs predominantly in the eastern portion of the deposit where it is tightly infolded with the felsic tuff unit. The mafic metavolcanics may locally be stratigraphically intercalated with the felsic tuffs. The rocks consist of alternating sequences of mafic flows, intermediate to mafic tuffs and mafic dikes. The flows are generally non-schistose, fine grained, massive, locally feldspar phyric, weakly to moderately magnetic, and dark-grey to black in colour. Locally observed vesicular and hyalopilitic zones have been interpreted as evidence for a flow origin for the bulk of the unit.

The mafic dikes that are included in this unit are very fine grained and slightly chloritized along their margins. The dike contacts are slightly oblique to schistosity. They are deformed, indicating emplacement early in the geological evolution of the deposit and they are thought to represent synvolcanic dikes and sills petrogenetically related to mafic flows. The dikes have very poor lateral continuity.

The mafic metavolcanic unit may represent part of a bimodal volcanic complex that could include the felsic extrusive units or they may be part of an older crustal section upon which the felsic tuff and the Lapilli tuff would have been deposited. Ongoing geochemical studies should provide more information on the petrogenetic origins of the different metavolcanic units.

Felsic Tuff

The felsic tuff unit consists predominantly of rhyolitic to dacitic rocks many of which preserve a fine lamination that suggests an origin as pyroclastic deposits. It is likely that rhyolitic and dacitic flows also make up a significant proportion of the unit. The groundmass is essentially quartz, feldspar and sericite. The rock is light grey in colour and it is generally strongly foliated. Quartz phenocrysts represent up to 10% and they often preserve euhedral bipyramidal shapes. The phenocrysts are embedded in a holocrystalline matrix of fine-grained quartz-feldspar-biotite-(sericite-chlorite). Primary magnetite is often lacking. Locally the quartz phenocrysts are flattened and stretched, with a distinctive blue tint. Pressure shadows at the tips of the deformed phenocrysts may be filled with fibrous quartz and/or sulphide minerals, principally pyrite.

Over 50% of the mineralization at the Montagne d'Or prospect is hosted in the felsic tuff unit.

Lapilli tuff

The Lapilli tuff unit consists of rocks of similar composition to the felsic tuff unit but with quartzo-feldspathic masses (lapilli) hosted within the rhyolitic to dacitic rock matrix. The bulk of the Lapilli tuff unit occurs in the southern part of the Montagne d'Or deposit, close to the sheared contact with the metasedimentary unit. Franklin (1999) suggested that a "felsic lapilli tuff" unit would represent a coarse basal sequence of an ash flow tuff sequence.

Granodiorite

The Granodiorite unit is composed of variably deformed, medium to coarse grained rock the main constituents of which are quartz-feldspar-biotite. Much of the unit is more or less equigranular although sub-rounded quartz and euhedral feldspar phenocrysts are common and are sometimes enclosed within a finer grained groundmass giving a porphyritic texture. The rock is light gray but locally is has a gray to cream colour due to sericitization and possibly some albitic and silicic alteration as well. Where the rock is strongly altered the primary texture is largely obliterated.

Quartz-feldspar Porphyry

This unit has a mineralogy that is similar to the Granodiorite unit from which it differs in colour and texture. The Quartz-feldspar porphyry is light gray to white and contains a large proportion of euhedral to subhedral phenocrysts of both quartz and feldspar. This unit might be a porphyritic facies of the Granodiorite unit; however, it tends to form homogeneous intervals of several metres in drill core and it is here assigned to its own unit.

Feldspar Porphyry

The Feldspar porphyry unit forms two dikes that are documented to cross-cut the Mafic volcanic, Felsic tuff and Granodiorite units. The rock is of intermediate to felsic composition with a dark grey colour and abundant, euhedral to subhedral feldspar phenocrysts. The rock can also contain a small proportion of blue quartz phenocrysts locally. The texture is invariably porphyritic and it can be strongly sheared, suggesting the dikes may have been emplaced within active shear zones.

Quartz-carbonate Veins

Quartz-carbonate veins vary in thickness from the millimetre to metre scales. They are observed to cross cut the principal tectonic S1 foliation and are also deformed and folded, hence they are interpreted to have formed syn-orogenically. The veins are not generally associated with mineralization. Within mafic flows and intrusions, they occur as white, metre scale veins that cross-cut lithologic layering. The quartz veins within the felsic units are thin and are white or blue-grey in colour.

Alteration

Gold mineralization at the Montagne d'Or deposit is accompanied by pervasive alteration which includes sericite, secondary biotite (generally retrograded to chlorite) and secondary K-feldspar with locally associated quartz. Alteration products are the result of partial replacement of all lithologies due to reactions with the Fe and sulphide rich mineralizing fluids. The predominant additions to the rock geochemistry were sulphur and iron, as well as potassium, gold, and base metals, with a concomitant removal of sodium and calcium. The precipitation mechanism for gold was likely direct interaction of hydrothermal fluids with the country rocks.

Sericite is the dominant alteration phase in the shallower part of the drillholes, from approximately 40 to 120 m down-hole depth. It transitions into secondary biotite below 150 m. The most pervasive alteration is dominantly a phyllic assemblage. This includes quartz-sericite-pyrite and veinlet-controlled potassic assemblages of secondary biotite, and associated pervasive secondary K-feldspar. A less common, propylitic assemblage consists of chlorite-epidote-calcite. Veinlet assemblages include; quartz-pyrite-pyrrhotite-chalcopyrite, secondary biotite-pyrite-pyrrhotite, and magnetite-pyrrhotite-chalcopyrite-quartz-chlorite with minor amounts of red garnet. Chloritization occurs as a pervasive alteration of mafic units, and as millimetre-scale veinlets within felsic lithologies. The chlorite is Fe-rich, in contrast to Mg-rich chlorite typically associated with VMS type alteration. There is no documented correlation between chloritization and gold content. There is, however, a weak correlation between "hyperchlorite" zones and gold mineralization. The hyperchlorite zones are typically deficient in gold but commonly located adjacent to strongly auriferous zones. The prominent addition to the mafic rocks is Fe^{3+}, as well as gold. This is in part due to addition of sulphide, and perhaps to formation of Fe-rich chlorite. The addition of K_2O, as either sericite, secondary biotite, or secondary K-feldspar is also present. Alteration is typically strongest at the margins of the mineralized zones.

Chlorite alteration within mafic and intermediate units may include some secondary biotite. Zonation of peripheral Lead-Zinc (Pb-Zn) disposed about an Gold-Copper (Au-Cu) centre is also suggestive of a porphyry-type system. Late stage, narrow quartz veins are planar and cross cut the foliation and mineralized veinlets. They typically have a broad selvage of carbonate-chlorite alteration.

Hyperchlorite alteration zones at Montagne d'Or are composed of variably chloritized portions of nearly all lithologies. They occur predominantly in the mafic volcanic units, intermittently in the felsic units and rarely in mafic intrusive units. The mineralogical and textural characteristics of the zones are quite similar in both mafic and felsic units. The hyperchlorite alteration zones are composed of well foliated biotite (with incipient chlorite replacement), and locally contain a calc-silicate-rich assembly of actinolite, garnet, quartz, calcite-dolomite and magnetite + pyrite, chalcopyrite and pyrrhotite. The magnetite within this assemblage appears to be hydrothermal, and some magnetite rich intervals with sulphides can be highly auriferous. These zones are interpreted as reflecting primary mineralization as opposed to post-mineralization processes.

The edges of the felsic tuff unit are characterized by chlorite veining. Quartz phenocrysts are preserved while most of the primary textures are destroyed, particularly within central parts. Sulphide rich zones up to 50% can be associated with the chloritic alteration. Some rocks logged as mafic tuff may actually represent highly chloritized felsic lithologies. Visual discrimination of hydrothermal and metamorphic chlorite is very difficult.

Silicification is fairly pervasive in all volcanic units. Within the centre of the Montagne d'Or prospect, less silicified units tend to have a higher sulphide content.

Sericitization is a major and widespread alteration feature within the felsic units. It has been interpreted as a later overprinting alteration stage on an earlier secondary K-feldspar. There is no documented association between sericitic alteration and gold content. However, the early BRGM regional geochemistry showed that K and Ba are elevated proximal to faults and shear systems. This feature in time provided the pathfinder to the Montagne d'Or prospect gold mineralization.

Carbonate alteration occurs within felsic rocks as fine stringers and replacements. Within mafic units, calcite development is more pervasive, occurring as massive replacement within rhythmically banded tuffs, and as carbonate-chlorite or quartz-carbonate veinlets. It is difficult to separate the hydrothermal alteration carbonates from that derived by regional metamorphic processes. No correlation has been noted between carbonate alteration and gold content.

7.2.3 Structure

The Project area has experienced two distinct deformational events. The first involved ductile deformation during the Lower Proterozoic accretionary arc tectonism that formed the Guiana Shield. The second is a more brittle deformation event associated with the faulting within the NGT.

The first phase of regional deformation was associated with a regional northeast-southwest compression that led to the development of the pervasive S1 schistosity that strikes 080° to 100° and that dips steeply south. At the Montagne d'Or deposit, the average strike of S1 is 087° and the average dip is 69°S. This principal deformation event postdates mineralization as evidenced by the highly deformed sulphide fabric. However, at the Montagne d'Or, the crystallization of sulphides with pressure shadows at the tips of deformed phenocrysts indicates that some sulphide may have been remobilized during the tectonic event or that a second sulphide deposition event may have been syn-deformational.

Regionally, the development of the S1 schistosity was accompanied by Upper Greenschist Facies and Lower Amphibolite Facies metamorphism, locally associated with the emplacement of granitic plutons and migmatization. S1 is associated with the deformation event that resulted in the very tight to isoclinal folding of the Montagne d'Or deposit and also in the thrusting of the amphibolite unit over the deposit.

The principal, penetrative foliation in the Montagne d'Or deposit is defined by the flattened and stretched feldspar and quartz crystals, as well as by aligned biotite crystals and locally by highly stretched biotite schlieren, referred to by SOTRAPMAG personnel as "black flames". The biotite is often wholly or partially replaced by dark green chlorite.

A second fabric has been reported by SOTRAPMAG geologists and confirmed by TGC (2016a); it consists of locally developed kink bands that deform the principal foliation on centimetric to metric scales. No orientation data are available specifically for the kink bands.

Various small scale, quartz-carbonate veins are present in the drill core. The distribution of poles to these veins is nearly identical to the distribution of poles to the principal foliation. Visual inspection of drill core from mineralized zones reveals that most veins are transposed parallel or near-parallel to the foliation by intense shearing, folding and boudinage.

Ductile shear zones are shown by local development of mylonitic textures. Their orientation measurements were classified by way of visual inspection into three sets – a dominate ENE striking set with average orientation of 086.3/68.7, a minor NE striking set with an average orientation of 039.7/60.3, and a minor NW striking set with an average orientation of 141.9/54.9.

The first of the three sets is parallel to the principal foliation and it undoubtedly represents anomalously high strain (intense foliation development) that was recognized by the logging geologists. The other two sets have average orientations that do not correspond closely to structures modeled at the scale of the deposit for the present study. However, allowing for the small sample set and for possibly poorly orientated drill core, the minor NE striking shears might represent the SZ2 set of shear zones described later in this report, and the minor NW striking shears might correspond to one of the modeled brittle fault sets.

The second phase of regional deformation postdates the EDS sediments and is related to sinistral transcurrent tectonism, marking the contact between the NGT and PGB. As a result of the second deformation, the earlier S1 schistosity is locally crenulated. A weak S2 fabric is characterized by a spaced cleavage, which strikes 060°. At the Montagne d'Or deposit, late diabase dikes have a preferred strike orientation between 060° and 065°, sub-parallel to S2, suggesting they were emplaced with shears, faults or fractures that had formed during the transcurrent tectonic event.

Ductile Shear Zone Rocks

Ductile shear zones within the deposit are characterized by mylonitic textures indicative of shearing and dynamic crystallization under upper greenschist to lower amphibolite metamorphic conditions. Felsic tuff hosts deformed quartz porphyroclasts set in a very fine grained, intensely foliated matrix. Sulphide + quartz + carbonate veins with dark chloritic margins are transposed parallel to the foliation. The felsic Tuff also hosts feldspar porphyroclasts which are strongly stretched and partly dynamically recrystallized but their forms are still visible. The shear foliation (probable C-planes) are highlighted by schlieren of dark mica, partly retrogressed to chlorite. Close inspection of mylonitic rocks reveals that pressure shadows at the ends of porphyroclasts are composed of quartz, calcite and in some cases sulphide.

The mylonitic rocks within the deposit are strongly cohesive, in part due to silicification that accompanied mineralization. It should be noted that some examples were observed of mylonitic fabrics that appeared to have been reactivated under retrograde brittle-ductile conditions, rendering the shear zones locally fissile.

Other shear zone facies are present within the Hanging wall thrust which places a volcano-sedimentary package structurally above the deposit. Within the ductile thrust, the mafic metavolcanic rocks are strongly foliated and they host variable proportions of transposed quartz + carbonate + sulphide (mainly pyrrhotite) veins. The metasedimentary facies also preserve rootless intrafolial isoclinal folds. (TGC 2016a)

Brittle-ductile Shear Zone Rocks

A facies of brittle-ductile shear zone rock was documented in several drillholes within the Montagne d'Or deposit. It is typified by the presence of closely spaced narrow chlorite-carbonate rich mylonite zones, each a few centimetres thick and often sulphide bearing, and associated fractures that are also filled with chlorite and carbonate. Within the brittle-ductile shear zones but outside of the shears and fractures the rock is silicified, albitized and sericitized. Sulphide bearing quartz carbonate veins are not uncommon within the brittle-ductile shear zones and in close proximity to them. Observations of several drillholes revealed that brittle-ductile shear zones are quite common in proximity to the major shear zones. (TGC 2016a)

Cohesive Brittle Fault Rocks

These rocks are breccias, composed of angular to subrounded fragments set in a finer grained comminuted matrix that may be silicified, chloritized, carbonate rich or a combination thereof. Some breccias have formed within fault zones that have foliated cataclasite margins indicative of brittle-ductile shearing. Other examples of brecciated fault rocks have sharp boundaries with unfaulted wall rocks. (TGC 2016a)

Weakly Cohesive and Non-Cohesive Brittle Fault Rocks

The weakly cohesive fault rock is a breccia with rock fragments set in a matrix of comminuted rock and gouge. These are generally non-cohesive faults characterized by crushed fragments that are not embedded in any indurated matrix or gouge. Such non-cohesive faults may host gouge in situ, however it would have been washed out in the course of drilling. (TGC 2016a)

7.2.4 Structural Model

The structural model was prepared by Terracognita Geological Consulting (TGC) using Leapfrog® Geo software platform. It consists of 23 structures that were generated as 25 Leapfrog® Geo solids using the fault system functionality. The solids were also exported as dxf files for use on other software platforms.

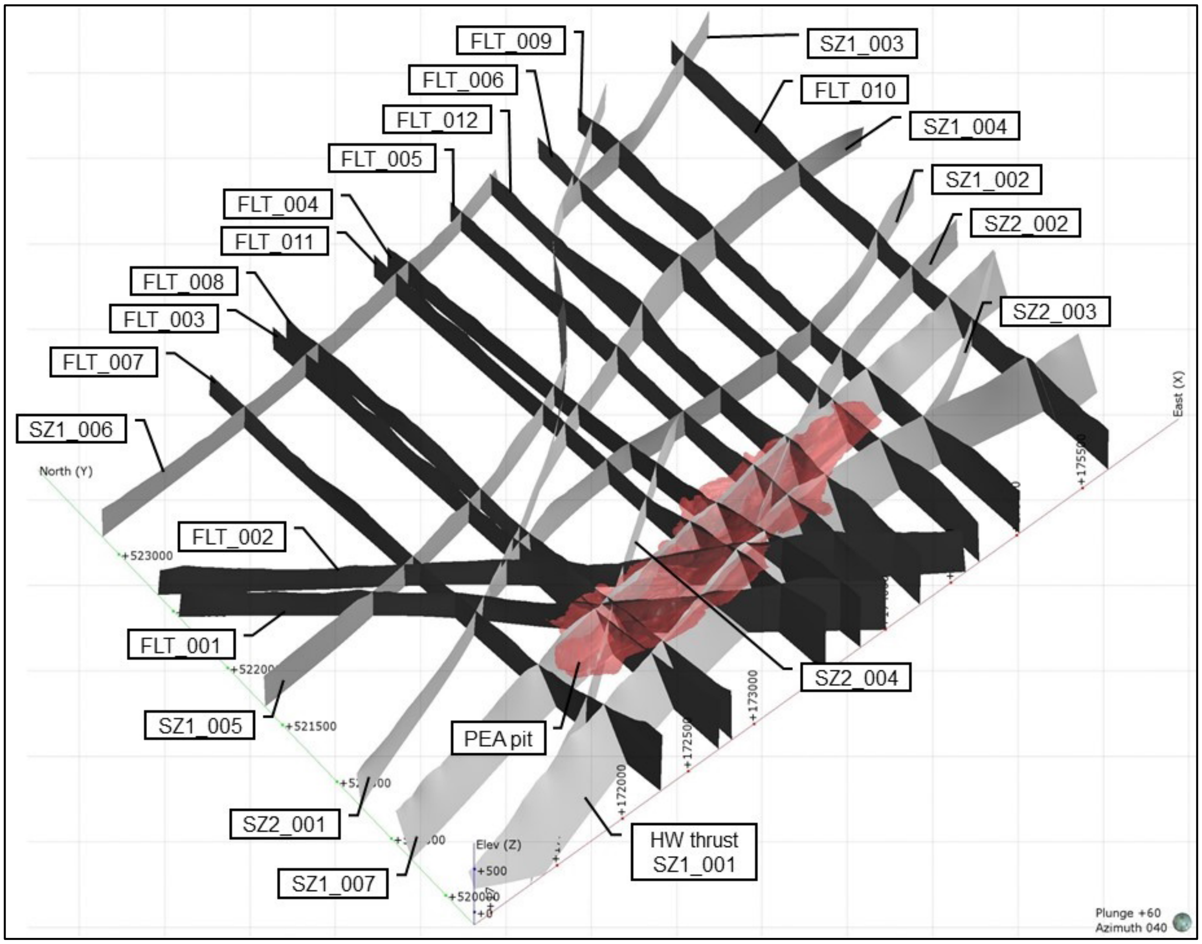
The modeled structures are listed in Table 7-1 and shown in Figure 7-5. The faults are numbered sequentially from FLT_001 through FLT_012. Shear zones are grouped into two sets and they are labeled accordingly – SZ1 structures are generally ENE to E-W striking, parallel to the overall structure of the Montagne d'Or deposit whereas SZ2 structures generally strike NE, 055° - 068°, counter-clockwise oblique to the SZ1 shear zone set. The individual SZ2 structures apparently merge asymptotically with SZ1 structures. It is likely that the SZ1 and SZ2 sets of shear zones would represent an anastomosing shear zone system.

Table 7-1: List of Geological Structures

STRUCTURE	COMMENTS
FLT_001	Average strike 130° ; Validated in MO14126 @ 63.66m, MO14169 @ 34.88m.
FLT_002	Average strike 128° ; Validated in MO9742 @ 138.17m, MO14173 @ 133.35m, MO14209 @ 169.55m.
FLT_003	Average strike 176° ; Validated in MO14187 @ 274.5m.
FLT_004	Average strike 173° ; Validated in 90.00mto MO12067 @ 264.00m, GT05 @ 90.00m.
FLT_005	Average strike 177° ; Validated in GT06 @ 109.48m, GT11 @ 183.20m.
FLT_006	Average strike 177° ; Validated in MO12077 @ 320.12m.
FLT_007	Average strike 175° ; Not intersected by drilling.
FLT_008	Average strike 176° ; Validated in MO9725 @ 163.50m.
FLT_009	Average strike 176° ; Not intersected by drilling; Should be intersected by Hydro3.
FLT_010	Average strike 177° ; Not intersected by drilling.
FLT_011	Average strike 175° ; Validated in MO14199 @ 188.2m, GT03 @ 184.75m, GT04 @ 120.3m.
FLT_012	Average strike 177° ; Not intersected by drilling.
SZ1_001_W SZ1_001_C SZ1_001_E	Ductile thrust cutting the package of mafic volcanics and metasedimentary rocks to the south of MDO, it is cross-cut and offset by FLT_004 and by FLT_010; W = West segment, tied to GT03 @ 81.40m - 119.73m; C = Central segment, tied to GT06 @ 195.00m; E = East segment.
SZ1_002 SZ1_003 SZ1_004 SZ1_005 SZ1_006 SZ1_007 SZ2_001 SZ2_002 SZ2_003 SZ2_004	These shear zones are interpreted by the author from the Lidar DEM and the Magnetic Tilt Angle map product. Several (apparently) retrograde type shears were observed in drill core along SZ2_004 and they may represent a shear zone facies associated with the NE striking SZ2 set of structures; see the text of this report for discussion of that facies. SZ1_007 was modeled to fit the Magnetic Tilt Angle anomalies and also the distribution of mineralization in the main mineralized zone (Upper Favorable Zone, UFZ).

FLT = Fault; SZ = Shear zone.

Source TGC 2016a



Source TGC 2016a

Figure 7-5: Perspective View of the Montagne d'Or Structural Model

Modeling Approach

As a first step, fault and shear zone map traces were interpreted using two principal data sets. The first is a high resolution Lidar-derived digital elevation mapping (DEM) which is a Columbus proprietary data set generated by La Société Altoa. The DEM was especially useful for identifying fault scarps and drainage that can follow fault controlled topographic lows. The second data set was a Magnetic Tilt Angle anomaly map generated Condor Consulting in 2014 by reprocessing the aeromagnetic data that were gathered by Geotech Airborne Ltd. on behalf of Golden Star Resources Ltd., in 2007. The Magnetic Tilt Angle, i.e. the tilt of the total gradient above the horizontal, tends to highlight shallow, high and low amplitude magnetic features. Linear magnetic lows were interpreted to indicate magnetite destruction during hydrothermal/metamorphic mineral reactions in shear zones. That interpretation was also applied by Condor in their analysis, however, the shear zones interpreted here do not necessarily correspond to those interpreted by Condor. Deflections and truncations of magnetic anomalies were also interpreted as indications of faults or shear zones.

Once map traces of the structures were established, the topographic and magnetic data was inspected for evidence of cross-cutting relations between structures. The shear zones appear to asymptotically merge, offsets would be expected across the modeled faults which certainly formed later than the shear zones. The only structural offsets that are identified in plan view are the offsets of the SZ1_001 structure, also referred to here as the Hanging wall thrust by Fault 004. One of the two offsets of that structure is shown in Figure 7-6.

The general lack of structural offsets as seen in plan view suggests that: 1) any strike-slip components of displacements on faults may be too small to be resolved in the data set, and/or 2) the main components of displacements on the faults may be vertical. In the latter case, little or no offset would occur in plan view where one steeply dipping or vertical structure is cross-cut by another.

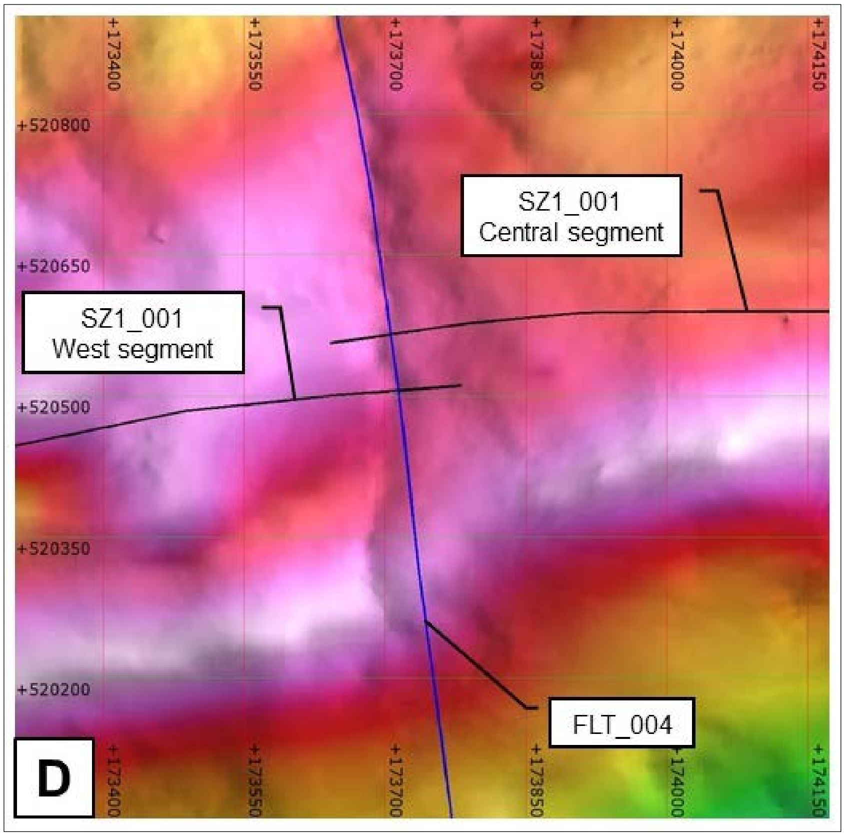
Eight of the 12 modeled faults were validated by direct observation in the drill core from at least one drillhole (Table 7-1) supporting the interpretations of fault traces based on topographic and geophysical data. The Hanging wall shear zone was validated in both drillholes, GT03 and GT06; the modeled base of the Hanging wall shear zone, SZ1_001, is placed at the base of the heavily sheared mafic metavolcanic rocks. Of the remaining shear zones, only SZ1_007, SZ2_002 and SZ2_004 cut through the heavily drilled volume of the deposit. The first of those three shear zones was not specifically validated in the inspected drillholes; the rocks in the mineralized zones were observed to be generally mylonitic so SZ1_007 was modeled to run through the main upper mineralized zone. In detail, the shear zones that appear to host the mineralized zones may be more complex. It is the opinion of TGC, that the SZ2 type structures, rather than being discrete, well defined shear zones, may in fact be shear "corridors" made up of numerous, possibly overlapping and linked brittle-ductile structures. During the review of drill core for the modeling exercise the brittle-ductile type of shears were only observed to cut the relatively coarse grained felsic intrusive rocks. It cannot be ruled out that the brittle-ductile structural facies could be a structural style limited to the felsic intrusive rock types in the Montagne d'Or deposit.

<u>Summary</u>

The structures modeled at the Montagne d'Or deposit include two sets of faults and two sets of shear zones. A total of twenty-three structures were modeled, 12 faults and 11 shear zones. All structures were initially interpreted based on a Lidar DEM and on airborne geophysical data products and then validated by inspection of drill core from the deposit. The modeling was performed using the Leapfrog® Geo software platform.

The two fault sets include one striking 130° and the other striking 175°. Both fault sets appear to have very steep dip angles, 88° to vertical. Cohesive and non-cohesive facies of fault rocks were documented from the modeled fault structures.

The SZ1 set of shear zones strikes 080° to 090° and the SZ2 set strikes 055° to 068°. Most of the shear zones appear to have very steep dip angles. Two of the SZ1 shear zones, SZ1_007 that would correspond to the main mineralized zone, and SZ1_001 representing the base of the Hanging Wall Thrust are modeled with dip angles between 68° and 72° to the south. Two shear zone facies are documented, ductile mylonites and brittle-ductile shear zone rocks. The mineralized zones are typically hosted by mylonites. The brittle-ductile shear zone facies may be typical of the deformation associated with the SZ2 shear zone set.



Source: TGC 2016a

Figure 7-6: Plan View Showing Offset of Magnetic High (Pink) Across Fault 004

7.3 Mineralization

The Montagne d'Or prospect consists of a family of tabular mineralized bodies that form closely-spaced sub-parallel east-northeast (084°) striking and steeply (72°) south-dipping mineralized zones. Mineralization has been encountered over a strike length of more than 2,500 m and to a vertical depth of at least 200 m. Only a small portion of the gold mineralization has been subjected to upper level oxidation. The significant fine-grained gold mineralization is principally affiliated with sulphide veins and masses within fresh country rock that begins at shallow depths.

Historically, on a macroscopic scale, two significant styles of gold mineralization have been recognized although they show a gradational relationship between each other:

- SMS with gold mineralization, and
- Sulphides in disseminated stringers with gold mineralization.

SMS was a term coined by previous operators and was used to support a "VMS" type model for the mineralization. It is characterized by a high sulphide content (>20%) and occurs over intervals ranging from tens of centimetres to up to 4 m. This mineralization was later interpreted to represent zones of thicker, deformed and transposed sulphide ± quartz-rich veins and a denser distribution of disseminated sulphide as compared to that of the disseminated type.

The SMS also includes sulphide-rich breccia dykes, which host rolled and milled clasts of host rock within a ductily deformed pyrite-chalcopyrite-pyrrhotite matrix. In addition, bornite is present, and minor amounts of arsenopyrite have been identified petrographically. There is a clear correlation between sulphide veinlets and sulphide-rich breccia zones and high gold grades. Relatively minor amounts of total sulphide (i.e., disseminated + vein and veinlet + breccia – hosted sulphide representing 2% to 5% total rock volume), locally resulting in significant although erratic, high gold concentrations, commonly attain values of tens of grams per tonne gold over standard 1 m sample intervals.

Disseminated mineralization is characterized by the presence of finely disseminated to finely fracture controlled sulphides, chiefly pyrite but with lessor and locally important chalcopyrite and pyrrhotite.

Close inspection of core and outcrop indicate that gold associated with this style of mineralization is in part controlled by the abundance of fine sulphide-quartz veinlets and fracture fillings which have been strongly (isoclinally) folded, sheared and transposed parallel to the S1 fabric. Grades for this mineralization type are dependent upon disseminated sulphide and sulphide-quartz veinlet density, but are generally low, in the 0.5 g/t Au to 3 g/t Au range over sample intervals which average approximately 1 m in length.

Mineralization is hosted by felsic, mafic and intercalated mafic/felsic rocks to varying degrees. However, approximately 80% of the gold mineralization in the deposit occurs within the more felsic units, mainly the Felsic tuff unit.

The mineralization appears as elongated lenses of higher grade material within broader zones of low grade but anomalous mineralization (0.25 g/t Au to 0.4 g/t Au). Several distinct anomalous mineralized domains are recognized, separated by barren intercalated mafic and felsic rocks.

Disseminated sulphide mineralization is hosted mainly within the Felsic tuff unit and is predominantly or entirely pre-orogenic. Disseminated pyrite crystals are coarse and also locally stretched. Some mafic units carry similar mineralization but with a notably lower sulphide vein density.

The Montagne d’Or deposit is now thought to be part of a stratiform/stratabound deposit type. Mineralization consists of pyrite, pyrrhotite and chalcopyrite with minor sphalerite, magnetite and arsenopyrite. Arsenopyrite, although observed, does not appear to have an obvious relationship with either gold or copper mineralization. Distinct phases are reported as stratiform disseminated sulphides, stockwork sulphide veinlets and layers of semi-massive sulphides that are tectonically transposed. The latter facies is considered as syn-volcanic in origin and as the most favourable occurrence for gold mineralization.

The disseminated sulphide veins could be related to feeder zones and/or remobilized on fold hinges and shear zones. In addition, evidence is found for tectonic remobilization with sulphides concentrated within fold hinges and pressure shadows, and cross-cutting sulphide-bearing veins.

Visible gold occurs in chlorite-rich zones or is spatially related to sulphide mineralization (after Giraud, Tremblay, Jébrak and Lefrançois, 2014). Figure 7-7 shows a photograph of native gold hosted by mafic volcanic rocks in drillhole MO1266 at a depth of 245 m. This particular 1 m interval ran 80.75 g/t Au. There is generally an increase in gold grades as sulphide (excluding pyrrhotite) content increases. Microscopic studies indicate that gold occurs as very fine grains in the host rock groundmass and at the junctions of quartz crystals. Gold has only very rarely been seen as inclusions within sulphide minerals.

Source: Columbus, 2013

Figure 7-7: Example of Visible Gold Occurring within Mafic Volcanics (MO1266)

8 Deposit Type

8.1 Mineral Deposit

The current interpretation is that Montagne d'Or is a deformed volcanogenic massive sulphide deposit (Ross 2014). Ross based this interpretation largely on the following details of the deposit.

- The presence of pillow basalts in the Upper Mafic Unit, making at least this part of the volcanic succession submarine, and formed on the ocean floor;
- The Felsic Unit is cut by tholeiitic mafic dikes related to the Upper Mafic Unit, whereas the Upper Mafic Unit is cut by calc-alkaline QFP dikes related to the Felsic Unit;
- This means that the Felsic Unit and the Upper Mafic Unit are broadly contemporaneous; by association, the Felsic Unit is therefore also submarine;
- The Felsic Unit is indeed, partly, a layered volcaniclastic pile (Franklin et al., 2001). There are some QFP intrusions in this pile (as noted by Shaw, 2001), but at least some of the felsic rocks were deposited on the sea floor (Franklin, 1999); volcaniclastic rocks are ideal for sub-seafloor replacement;
- Alteration mineralogy is dominated by sericite and chlorite, which are typical VMS minerals, or their metamorphosed equivalents (e.g., garnet, biotite); and
- The sulphides were emplaced before tectonic deformation.

A submarine volcanic arc is presently thought to be the likeliest setting for the formation of the Montagne d'Or deposit; the Izu-Bonin arc south of Japan may be a plausible analogue (there are seafloor massive sulphides deposits currently forming in this arc; e.g., Glasby et al., 2000). A back-arc with a strong subduction signature is also possible, as back-arc basins can have voluminous felsic magmatism too, for example the Manus Basin offshore Papua New Guinea, where there are also seafloor massive sulphides actively accumulating (e.g., Binns and Scott, 1993; Paulick et al., 2004; Ross, 2014).

9 Exploration

Since completing the previous technical report effective to the end of July 2015, Columbus has only conducted exploration drilling. The latest drilling program was completed in March 2016.

10 Drilling

Sections 10.1 and 10.2 have been excerpted from the Coffey 2014 Technical Report. Section 10.4 is updated current to this report. Standardizations have been made to suit the format of this report.

Since the inception of exploration by Columbus, a total of 171 drillholes (MO1361 to MO14231) have been completed testing the Montagne d'Or deposit.

Earlier drilling completed by Guyanor consists of a total of 56 drillholes (MO9601 to MO9856) totaling 10,916 m on from 1996 to 1998. Assays from these drillholes are of lower quality (a characteristic that has been taken into account during resource classification) but are considered as relevant and fit-for- purpose for the resource estimate. (note: all holes drilled by Columbus are within the deposit; however, there are three Guyanor holes, hole numbers MO57, MO58, and MO59, which were drilled in 2001 on the Apollon target located to the southeast of the deposit, and drillhole MO60, the only hole drilled in 2007, which is not included in the database as it is a twin of a previous hole).

10.1 Guyanor Drilling Program: 1996 to 1998

From 1996 to 1998, Guyanor completed a total of 56 drillholes (MO9601 to MO9856) totaling 10,916 m on the Montagne d'Or prospect. Drilling was done under contract by Major Drilling Company of Canada. Drill pads and access were prepared using bulldozers and/or excavators; every attempt was made to limit deforestation and for this reason, use of an excavator was preferred for construction of drill pads.

Drilling procedures were to collar each hole with HQ bits (core diameter 6.35 cm) and reduce to NQ (core diameter 4.76 cm) when hard and not oxidized rock was intersected. Core recovery was routinely measured and recorded for each core run. Core recoveries overall were generally excellent. Major Drilling used Longyear 38 wireline diamond drilling rigs. Drillhole spacing is variable, from 50 to 250 m. Drill fences are spaced 100 to 200 m apart. The presence of clearly visible, regionally consistent, and well-defined S_1 fabric allowed the core to be manually oriented in the core boxes, although local variations have, on occasion, caused incorrect orientation. Saprolite was not oriented due to the absence of a clearly defined fabric. Core was placed in plastic core boxes at the drill site, with core markers placed at the start and end of each core run, and boxes securely covered. Core boxes were transported back to camp for detailed logging and core splitting. Core photography was carried out infrequently. All drillhole collars were a surveyed for X, Y, Z coordinates tied to the mine grid shortly after completion so as to provide an accurate location for resource estimation. The mine grid was tied to the X, Y UTM grid and the Z coordinates were shifted 1,000 masl so that no negative elevations were present within the drillholes. Drillhole location surveys were performed by Guyanor survey crews and external surveyors from SATTAS using TDS equipment.

The first 47 drillholes were surveyed downhole for deviation and deflection by Major Drilling, mainly using acid bottle etch or Pajari /Tropari mechanical instruments. Downhole survey intervals were at 50 m. The final eight drillholes were surveyed in with Sperry Sun equipment. The downhole surveys using acid bottle etch and Tropari equipment were criticized within internal Guyanor documents as poorly suited to the task as only dip and no azimuth is recorded. The inaccuracy of the early downhole surveys is considered in Mineral Resource classification although it should be noted that due to the relatively short length, significant drillhole deviation and deflection at Montagne d'Or are minimal, with deflection of 5° to 10° over 200 m typical. Four drillholes were not collar surveyed; however, the planned hole coordinates have been used. Details for the drilling completed by Guyanor from 1996 to 1998 (56 holes in total) are listed in Table 10-1.

Table 10-1: Drillholes Completed by Guyanor from 1996 to 1998

Drillhole	Easting	Northing	Elevation	Azimuth	Dip	Depth (m)	Operator	Year
MO9601	173091.8	520520.8	260.89	0	-60	199.8	Guyanor	1996
MO9602	173096.5	520499.6	268.60	0	-60	52.5	Guyanor	1996
MO9603	173051.7	520634.9	220.88	0	-57	271.6	Guyanor	1996
MO9604	173311.3	520611.1	269.68	0	-61	208.6	Guyanor	1996
MO9605	173298.7	520711.7	229.45	0	-61	201.3	Guyanor	1996
MO9606	173706.1	520583.6	273.74	0	-60	199.6	Guyanor	1996
MO9607	173717.2	520708.8	258.57	0	-60	202.6	Guyanor	1996
MO9608	173703.7	520765.8	227.79	0	-60	199.6	Guyanor	1996
MO9609	173703.5	520873.9	180.19	0	-60	199.6	Guyanor	1996
MO9610	173331.9	520908.4	173.50	0	-60	199.6	Guyanor	1996
MO9611	173302.2	520802.4	191.95	0	-63	201.6	Guyanor	1996
MO9612	173014.3	520820.4	163.98	0	-61	201.55	Guyanor	1996
MO9613	172973.3	520738.8	182.53	0	-60	59.7	Guyanor	1996
MO9614	172969.8	520742.4	182.30	358	-61	205.6	Guyanor	1996
MO9615	172763.0	520800.2	186.52	0	-59	193.6	Guyanor	1996
MO9616	172730.8	520700.8	189.25	0	-60	199.6	Guyanor	1996
MO9617	173335.5	521128.7	120.17	0	-60	151.6	Guyanor	1996
MO9618	173312.4	521000.7	151.73	0	-60	156.6	Guyanor	1996
MO9719	174129.7	520732.4	296.82	0	-60	199.5	Guyanor	1997
MO9720	174136.4	520822.6	247.89	0	-60	200	Guyanor	1997
MO9721	173540.1	520678.7	273.57	0	-60	200	Guyanor	1997
MO9722	173534.4	520755.3	237.60	0	-60	199.5	Guyanor	1997
MO9723	172233.0	520519.2	233.78	0	-60	199.5	Guyanor	1997
MO9724	172236.9	520619.4	219.33	0	-60	198.5	Guyanor	1997
MO9725	172766.0	520594.6	228.71	0	-60	199.5	Guyanor	1997
MO9726	174626.3	520774.4	204.39	0	-60	199.5	Guyanor	1997
MO9727	174619.1	520860.7	184.97	0	-60	199.5	Guyanor	1997
MO9728	174225.2	520750.7	300.96	0	-60	199.5	Guyanor	1997
MO9729	172337.5	520852.9	172.76	0	-60	202.6	Guyanor	1997
MO9730	172441.8	520929.5	141.84	0	-60	199.6	Guyanor	1997
MO9731	172897.2	520696.3	208.28	0	-60	199.6	Guyanor	1997
MO9732	172819.0	520493.9	251.18	0	-60	277.6	Guyanor	1997
MO9733	172601.1	520591.9	231.09	0	-60	199.6	Guyanor	1997
MO9734	173522.6	520581.7	321.30	0	-60	22.7	Guyanor	1997
MO9735	173528.4	520578.9	321.39	1	-61	295.6	Guyanor	1997
MO9736	173919.9	520736.5	285.07	0	-60	199.6	Guyanor	1997
MO9737	174222.9	520641.5	298.81	0	-60	271.6	Guyanor	1997
MO9738	174430.2	520753.1	262.39	0	-60	263.9	Guyanor	1997
MO9739	174627.0	520672.8	218.79	0	-60	249.6	Guyanor	1997
MO9740	172969.6	520672.7	227.21	0	-59.5	229.6	Guyanor	1997
MO9741	173051.5	520732.8	177.60	0	-60	196.6	Guyanor	1997
MO9742	173013.2	520736	179.61	358	-60	190.6	Guyanor	1997
MO9743	174806.0	520885	203.38	0	-60	187.6	Guyanor	1997
MO9744	174808.3	520780.7	209.90	0	-60	199.6	Guyanor	1997
MO9745	175107.2	520887.9	193.47	0	-60	193.6	Guyanor	1997
MO9746	175107.0	520788.2	203.67	0	-60	238.4	Guyanor	1997
MO9747	175479.7	520760.9	184.40	90	-60	120.06	Guyanor	1997
MO9748	175479.7	520760.9	184.40	0	-60	193.6	Guyanor	1997
MO9849	172826.3	520709.3	205.80	0	-60	178.6	Guyanor	1998
MO9850	174331.3	520751.2	296.62	0	-60	150.9	Guyanor	1998
MO9851	174025.5	520755.9	277.46	0	-60	199.6	Guyanor	1998
MO9852	173923.2	520780.7	266.02	0	-60	151.6	Guyanor	1998
MO9853	173834.6	520751.4	257.74	0	-60	190.6	Guyanor	1998
MO9854	172895.1	520592.8	260.41	0	-60	199.6	Guyanor	1998
MO9855	173975.4	520754.1	277.44	0	-60	202.6	Guyanor	1998
MO9856	174075.4	520762.1	270.31	0	-60	211.6	Guyanor	1998

Source: Coffey, 2014
Coordinate System: CSG 167 datum UTM Zone 22

10.2 Columbus Drilling Program: 2011 to 2012

From the end of 2011 until August 2012, Columbus drilled 45 drillholes (MO11061 to MO12105) totaling 15721.45 m, named as Phase I of Columbus drilling. Drilling was done under contract by Performax Drilling of Val d'Or, Quebec, Canada.

Drilling procedures were very similar to those in the previous dill programs. All drillholes were collared using HQ equipment, downsizing to NQ after intersecting solid generally un-oxidized rock. Core recovery at the drill site averages 87.5% in HQ core (saprolite zone) increasing to 99.6% in NQ core (fresh material). Performax used a containerized Longyear 38 drill.

The drill program was designed to provide infill drillholes in known mineralized areas and to continue exploring strike extensions of the mineralization. Drillhole spacing in the central part of the mineralized zone varies between about 35 and 75 m and 100 to 200 m on the extremities.

The drillholes are, in general, inclined moderately to the north whereas the mineralization dips at 68° to 72° to the south. Therefore, the drillholes intercepts do not represent true thickness but true thickness averages approximately 75% of the intercept distance. Down-hole surveying of the drillholes was performed by the drill crew using a Reflex instrument. In some cases the Reflex instrument did not function correctly. For these holes an average was taken of measurements from 10 holes and these values were used where data could not be measured. Given that the deviation in all of the drillholes is very consistent this method is considered acceptable with minimal risk to the resource estimate.

A private contractor was hired to undertake surveying of all collars for holes MO1161 to MO11105 using CGS1967 datum. All drillhole collars were surveyed using GPS Total Station equipment. All previous drillhole coordinates were converted to CGS 1967 format, the 1,000 m elevation addition removed that was present in the earlier data and four older drill collars checked by re-surveying.

Details for the drilling completed by Columbus from 2011 to 2012 (45 in total) are provided in Table 10-2.

Table 10-2: Drillholes Completed by Columbus in 2011 and 2012

Drillhole	Easting	Northing	Elevation	Azimuth	Dip	Depth (m)	Operator	Year
MO11061	173870.5	520648.2	299.21	0	-70	350	Columbus	2011
MO11062	173984.3	520647.0	318.16	0	-70	399.3	Columbus	2011
MO11063	174072.9	520652.0	323.75	2	-60	378.5	Columbus	2011
MO11064	172972.3	520539.8	287.46	2	-60	419	Columbus	2011
MO11065	172891.8	520514.1	272.39	0	-60	356	Columbus	2011
MO12066	173770.8	520701.9	260.19	0	-60	329	Columbus	2012
MO12067	173637.9	520647.7	267.48	0	-60	361	Columbus	2012
MO12068	173441.4	520625.4	302.83	0	-60	380	Columbus	2012
MO12069	172745.1	520503.5	256.82	0	-60	257	Columbus	2012
MO12070	173025.8	520633.6	231.30	0	-60	302	Columbus	2012
MO12071	173206.0	520874.6	177.80	180	-50	308	Columbus	2012
MO12072	173057.3	520786.8	182.46	180	-50	350	Columbus	2012
MO12073	172615.6	520814.2	197.01	180	-50	440	Columbus	2012
MO12074	174676.1	520781.4	214.08	0	-60	275	Columbus	2012
MO12075	174516.8	520766.1	220.82	0	-60	251	Columbus	2012
MO12076	174435.1	520938.4	197.49	180	-50	322	Columbus	2012
MO12077	174641.4	520982.6	175.94	180	-50	429	Columbus	2012
MO12078	173868.8	520909.9	204.46	180	-50	411	Columbus	2012
MO12079	173647.8	520914.1	180.94	180	-50	375	Columbus	2012
MO12080	173438.0	520852.2	203.50	180	-50	387	Columbus	2012
MO12081	174275.9	520736.9	306.87	0	-60	345	Columbus	2012
MO12082	174168.4	520723.3	307.91	0	-60	351	Columbus	2012
MO12083	174377.1	520732.0	282.60	0	-60	317	Columbus	2012
MO12084	174383.6	520739.2	282.61	180	-50	152	Columbus	2012
MO12085	174131.7	520647.2	332.27	0	-60	425	Columbus	2012
MO12086	174177.0	520640.8	324.31	0	-60	425	Columbus	2012
MO12087	173436.6	520764.9	239.73	0	-60	302	Columbus	2012
MO12088	173485.4	520764.4	247.61	0	-60	299	Columbus	2012
MO12089	173586.3	520732.8	244.44	0	-60	299	Columbus	2012
MO12090	173303.8	520552.2	287.75	0	-60	409	Columbus	2012
MO12091	173220.9	520589.5	273.50	0	-60	400	Columbus	2012
MO12092	173022.7	520529.7	286.08	0	-60	374	Columbus	2012
MO12093	172924.8	520529.8	281.68	0	-60	448	Columbus	2012
MO12094	173101.5	520495.5	269.14	0	-60	464	Columbus	2012
MO12095	172845.4	520562.1	264.26	0	-60	365	Columbus	2012
MO12096	172604.5	520508.4	237.74	180	-60	119	Columbus	2012
MO12097	172603.7	520503.0	238.06	0	-60	422	Columbus	2012
MO12098	172636.2	520437.3	239.94	0	-60	389	Columbus	2012
MO12099	172423.6	520558.3	301.23	0	-60	221	Columbus	2012
MO12100	173169.6	520544.9	282.64	0	-60	381	Columbus	2012
MO12101	173261.1	520557.3	283.30	0	-50	350	Columbus	2012
MO12102	173363.8	520634.2	274.71	0	-60	344	Columbus	2012
MO12103	173394.2	520670.0	272.00	0	-60	281	Columbus	2012
MO12104	173490.1	520704.8	273.95	0	-70	346.65	Columbus	2012
MO12105	173587.3	520673.7	273.08	0	-60	413	Columbus	2012

Source: Coffey, 2014
Coordinate System: CSG 167 datum UTM Zone 22

10.3 Columbus Drilling Program: 2013 to 2014

From early 2013 until November 2014, Columbus drilled a total of 126 drillholes (MO13106 to MO14231) (25,073.6 m) and 13 abandoned and re-drilled holes (495.0 m), for a total of 25,568.6 m. This corresponds to the Phase II of Columbus drilling. Drilling was done under contract by Performax Drilling of Val d'Or, Quebec, Canada. Drilling procedures were the same as those in the previous programs. All drillholes were collared using HQ equipment downsizing to NQ after intersecting solid generally un-oxidized rock. Core recovery at the drill site averages 87.5% for HQ drillholes in the saprolite zone and 99.6% in NQ drillholes in fresh material. Details of the most recent drillholes completed by Columbus in 2013 and 2014 are presented in Table 10-3.

Table 10-3: Drillholes Completed by Columbus in 2013 and 2014

Drillhole #	UTM East	UTM North	Elevation (m)	Azimuth	Dip	Length (m)
MO14113	173220	520780	240	0	-60	118
MO14114	173170	520800	195	0	-60	88.6
MO14115	173170	520745	215	0	-60	167
MO14116	173350	520745	230	0	-60	149
MO14117	173260	520665	250	0	-60	161
MO14118	173100	520770	190	0	-60	74
MO14119	173100	520725	190	0	-60	143
MO14120	172930	520760	175	0	-60	122
MO14121	172890	520770	175	0	-60	110.5
MO14122	172810	520765	190	0	-60	111.5
MO14123	172750	520760	180	0	-60	104
MO14124	172700	520760	180	0	-60	101
MO14125	172650	520750	200	0	-60	124
MO14126	172600	520740	210	0	-60	122
MO14127	172500	520760	210	0	-60	98
MO14128	172650	520630	200	0	-60	123
MO14129	173775	520860	200	0	-60	122
MO14130	173875	520820	235	0	-60	98
MO14131	173825	520815	220	0	-60	98
MO14132	173925	520840	240	0	-60	107
MO14133	173975	520835	250	0	-60	121.5
MO14134	174025	520850	235	0	-60	111
MO14135	174075	520840	235	0	-60	131
MO14136	174175	520865	240	0	-60	101
MO14137	174225	520840	255	0	-60	164
MO14138	173590	520865	200	0	-60	116
MO14139	173540	520870	200	0	-60	95
MO14140	174575	520850	190	0	-60	134
MO14141	174675	520895	200	0	-60	119
MO14142	174675	520840	210	0	-60	179
MO14143	174525	520830	215	0	-60	169
MO14144	174475	520850	215	0	-60	158
MO14145	174375	520865	235	0	-60	131
MO14146	174425	520840	230	0	-60	155
MO14147	174525	520880	195	0	-60	101
MO14148	173010	520465	290	0	-60	150.8
MO14149	172850	520630	245	0	-60	164
MO14150	172810	520620	245	0	-60	161
MO14151	172400	520620	275	0	-60	125
MO14152	172500	520600	275	0	-60	161
MO14153	172707	520584	215	0	-52	159.7
MO14154	172650	520730	200	0	-60	155
MO14155	172600	520700	215	0	-60	173
MO14156	172400	520700	230	0	-60	149
MO14157	172500	520700	230	0	-60	184
MO14158A	172700	520720	185	0	-60	22.5

Drillhole #	UTM East	UTM North	Elevation (m)	Azimuth	Dip	Length (m)
MO14158	172700	520720	185	0	-60	153
MO14159	172850	520750	190	0	-60	130
MO14160	173400	520795	220	0	-60	166
MO14161	173700	520835	195	0	-60	158
MO14162	174326	520829	254	0	-60	198
MO14163	174276	520839	247	0	-60	171
MO14164	172399	520510	304	0	-60	266
MO14165	172499	520540	292	0	-60	221
MO14166	172809	520570	249	0	-60	226.6
MO14167	172969	520620	253	0	-60	191
MO14168	172930	520605	270	0	-60	242
MO14169	172849	520515	263	0	-60	287.6
MO14170	172930	520670	225	0	-60	281
MO14171	173051	520568	256	0	-60	257
MO14172	173099	520595	244	0	-60	200
MO14173	173099	520665	215	0	-60	260
MO14174	174025	520695	303	0	-62	316.9
MO14175	173875	520695	290	0	-62	278
MO14176	174025	520620	330	0	-62	365
MO14177	173925	520680	302	0	-62	317
MO14178	173975	520680	307	0	-62	323
MO14179	173925	520620	318	0	-62	329
MO14180A	172969	520500	292	0	-62	125
MO14180	172969	520500	292	0	-62	344
MO14181	173825	520620	299	0	-62	307
MO14182	173775	520575	308	0	-62	344
MO14183A	173775	520640	283	0	-62	98
MO14183B	173775	520640	283	0	-62	15.5
MO14183	173775	520640	283	0	-62	266
MO14184	173700	520660	259	0	-62	293
MO14185	173825	520690	273	0	-62	349
MO14186	172699	520540	230	2	-62	230
MO14187	172650	520600	205	0	-60	299
MO14188	172550	520760	210	0	-60	104
MO14189	172400	520735	219	0	-60	145
MO14190	173875	520765	260	0	-62	177
MO14191A	172499	520490	286	0	-62	62
MO14191	172499	520490	286	2	-62	301
MO14192	172650	520535	220	2	-62	230
MO14193	172550	520610	250	0	-60	308
MO14194	172550	520550	265	0	-60	239
MO14195	172929	520490	273	0	-64	322.8
MO14196	172889	520465	261	0	-62	108
MO14197	172849	520455	259	0	-62	123
MO14198	173650	520590	276	1	-64	320
MO14199A	173440	520675	289	2	-65	30.5
MO14199B	173440	520675	289	2	-65	36.5
MO14199	173440	520675	289	2	-65	320
MO14200	173169	520680	225	1	-63	239
MO14201	173590	520600	301	2	-65	353
MO14206	172550	520700	225	0	-62	191
MO14207	173169	520490	287	0	-62	188
MO14208	173219	520545	293	0	-64	353
MO14209A	173169	520605	264	0.5	-62	15.5
MO14209	173169	520605	264	0.5	-62	247
MO14210	173220	520700	235	0	-62	197

Drillhole #	UTM East	UTM North	Elevation (m)	Azimuth	Dip	Length (m)
MO14211	173650	520710	242	1	-64	293
MO14212	173590	520810	212	1	-63	182
MO14213	173650	520790	199	0	-63	179
MO14214	173825	520815	225	2	-65	194
MO14215A	174225	520700	317	2	-65	18.5
MO14215	174225	520700	317	2	-65	251
MO14216	174576	520784	202	2	-63	210
MO14217	174576	520726	212	1	-63	270
MO14218A	173010	520565	276	2	-63	18.5
MO14218	173010	520565	276	2	-63	269
MO14219	174476	520725	245	1	-63	182
MO14220	174276	520650	286	2	-63	257
MO14221	174275	520690	301	1	-63	227
MO14222	174175	520800	270	2	-64	188
MO14223	174375	520800	265	1	-63	233
MO14224	174476	520774	239	1	-62	239
MO14225A	174326	520705	294	0	-63	6.5
MO14225	174326	520705	294	0	-63	206
MO14226	174526	520716	212	1	-63	280
MO14227A	174376	520685	275	1	-63	15.5
MO14227	174376	520685	275	1	-63	224
MO14228A	174426	520710	258	1	-63	30.5
MO14228	174426	520710	258	1	-63	200
MO14229	174675	520723	216	1	-64	290
MO14230	172450	520590	275	1	-62	143
MO14231	172450	520530	306	1	-63	233
Total Metres						**25,568.6**

Source: Columbus, 2015

10.4 Columbus Drilling Program: 2015 to 2016

From October 2015 until March 2016, Columbus drilled a total of 63 drillholes totaling 6,297.1 m). This corresponds to the Phase III of Columbus drilling. Drilling was done under contract by Au Drilling based in Goedverwagting, Guyane and Pro Forage Drilling based in Cayenne, French Guiana. Drilling procedures were the same to those in the previous programs with the exception that some reverse circulation drilling was used in the Phase III program. Au Drilling completed 31 RC holes and 4 holes with rotary circulation (RC) tops with diamond core tails. RC drilling was only used until water was encountered at that depth the hole was terminated or a core tail was completed. All drillholes were collared using HQ equipment downsizing to NQ after intersecting solid generally un-oxidized rock. Core recovery at the drill site averages 87.5% for HQ drillholes in the saprolite zone and 99.6% in NQ drillholes in fresh material. Details of the most recent drillholes completed by Columbus in Phase III are presented in Table 10-4.

Table 10-4: Drillholes Completed by Columbus in 2015 and 2016

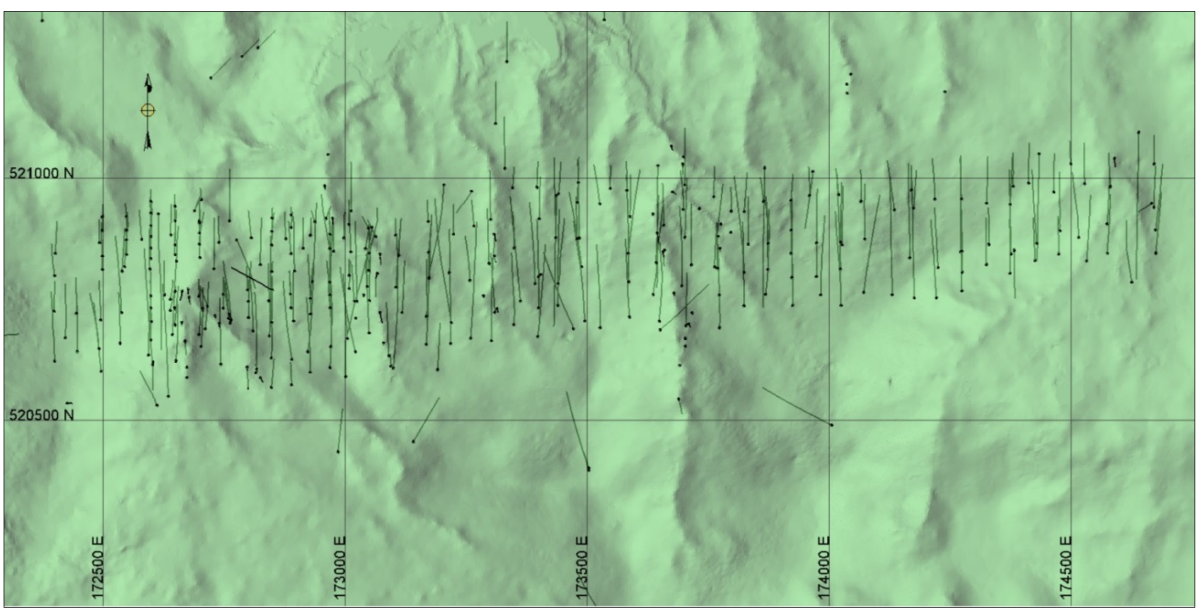
Drillhole #	UTM East	UTM North	Elevation (m)	Azimuth	Dip	Length (m)
MO15240DD	172598.348	520880.342	207.618553	0	-60	131.15
MO15241DD	172498.347	520839.352	220.5921479	0	-60	141.6
MO15242DD	172548.348	520842.332	215.7165691	0	-60	140.94
MO15243DD	172544.348	520817.332	221.2803432	0	-60	175.5
MO15264DD	172698.3885	520911.383	175.12368	2	-60	88.73
MO15265DD	172688.6684	520932.8331	171.9963006	20	-57	60.15
MO15281DD	172648.3381	520942.323	180.9832268	0	-60	50.1
MO15283DD	172648.368	520842.342	200.755462	0	-60	191.98
MO15284DD	172598.348	520856.362	207.6627362	0	-60	160.15
MO16246DD	172598.369	520666.321	243.3751633	0	-60	176.25
MO16247DD	172593.35	520635.311	245.0401094	0	-59	207.05
MO16248DD	172648.369	520697.391	213.0850777	0	-60	152.3
MO16249DD	172698.359	520677.331	223.1205594	0	-58	167.55
MO16250DD	172848.32	520702.302	260.6522981	0	-60	184.02
MO16251DD	172928.351	520751.363	242.5773012	0	-62	150.5
MO16252DD	173048.391	520809.314	196.7442205	0	-58	66.14
MO16253DD	172798.36	520769.322	233.9217536	0	-58	96.99
MO16254DD	173045.382	520776.313	213.6050164	0	-58	122
MO16255DD	173008.361	520799.353	212.3914516	0	-60	66.13
MO16256DD	173008.361	520772.343	225.7589049	0	-60	106.82
MO16257DD	172968.341	520758.393	240.314992	0	-60	128.15
MO16258DD	172888.361	520732.432	249.2620329	0	-60	150.73
MO16259DD	172848.35	520807.333	214.3293409	0	-60	45.5
MO16285DD	172598.328	520834.342	209.2732931	0	-60	186.82
MO16290DD	172748.399	520760.372	225.0059424	0	-55	65
MO16291DD	172762.71	520701.6	230.77972	350	-63	161.47
MO16292DD	172748.4	520732.322	229.1339655	0	-55	100.76
MO16293DD	172698.239	520756.362	202.9844846	0	-55	60.9
MO16294DD	172715.75	520736.48	212.1331301	340	-65	103
Total Core						**3,638.38**
MO15263RD	172698.3587	520864.3327	181.637174	2	-60	151
MO15272RD	172888.3101	520844.3532	193.9366107	354	-60	134.95
MO15282RD	172648.3784	520862.7125	197.9884767	0	-60	150.2
MO15286RD	172648.388	520912.273	192.1869508	0	-56	94
Total Core and RC						**530.15**
MO15233RC	172598.3377	520952.3229	193.3276339	0	-60	50
MO15234RC	172598.3378	520928.3328	199.730373	357	-60	65
MO15235RC	172548.3574	520921.4025	195.5177186	356	-60	55
MO15236RC	172498.3471	520920.3723	189.1291432	3	-60	55
MO15237RC	172498.3572	520892.3821	201.7643938	4	-60	65
MO15238RC	172548.3475	520892.3923	207.5716037	3	-60	75
MO15239RC	172598.3479	520902.3626	206.2613215	2	-60	100
MO15240RC	172598.3479	520880.3424	207.6185651	0	-60	130
MO15244RC	172598.3384	520757.4015	223.5838404	0	-60	63
MO15245RC	172598.3385	520732.3813	226.8825458	0	-60	90
MO15260RC	172638.3788	520727.4315	204.5145709	13	-60	91
MO15261RC	172638.4187	520750.3216	201.6579496	13	-61	72
MO15262RC	172648.3987	520767.3418	200.9253132	0	-60	40
MO15264ARC	172698.3885	520910.383	175.3211784	1	-60	90
MO15264RC	172698.3886	520907.383	175.9200683	3	-61	48
MO15265RC	172688.6684	520932.8331	171.9963006	24	-56	48
MO15266RC	172703.2984	520956.2833	170.8939282	0	-60	48
MO15267RC	172848.3596	520918.3636	168.8869343	5	-60	45
MO15268RC	172848.3597	520890.3434	178.5500212	3	-60	70
MO15269RC	172888.3599	520898.3536	172.2488794	4	-60	65
MO15270RC	172937.3303	520910.2139	160.6646224	343	-60	50
MO15271RC	172888.3599	520924.3038	162.0410538	2.2	-60	35.6

Drillhole #	UTM East	UTM North	Elevation (m)	Azimuth	Dip	Length (m)
MO15273RC	173041.9512	520854.6338	172.4416618	9.7	-56	90
MO15274RC	172997.1908	520876.4538	166.3327252	0	-54	90
MO15275RC	172968.3806	520884.3338	167.3062038	0	-60	99
MO15276RC	172960.3604	520917.204	152.8965098	21	-60	55
MO15277RC	172928.3803	520882.3236	169.1094374	3	-61	78
MO15278RC	173063.3813	520880.2941	181.2473345	334	-58	70
MO15279RC	173048.3811	520897.3142	181.0298041	332	-54	50
MO15280RC	173008.3708	520905.3741	163.10196	0	-60	50
MO15283RC	172648.368	520842.342	200.755462	0	-60	96
Total RC						**2,128.6**
Grand Total						**6,297.13**

10.5 Interpretation of Drillhole Results

The drilling types described above all constitute industry standard methods of exploration for this type of mineralization and material. The sampling procedures all meet industry best practices and an appropriate chain of custody has been utilized during all handling and sampling of the drill core or cuttings. The drillholes are inclined on average at -60° toward the -70° dipping mineralization; therefore, the drillhole intersections do not represent true thickness of the mineralization. The drillholes generally intersect the mineralization at approximately 50°, which SRK considers appropriate to define the geologic model and mineralization.

SRK is of the opinion that best professional judgment, and appropriate exploration and scientific methods were utilized in the collection and interpretation of the drilling data used in this report. The sampling is sufficient and spaced appropriately to support the resource estimation. Figure 10-1 presents an overview of the drillhole locations.



Source: SRK, 2016

Figure 10-1: Plan View of Drillhole Traces

11 Sample Preparation, Analysis and Security

The information presented in this section concerning pre-2011 sampling and analysis has been largely based on the SRK report, (Stryhas, 2012) with additional information as specified for updated drilling data provided by Columbus.

11.1 Historical Methods

Limited information is available on the historical transport, sampling and analysis of the Guyanor drillholes. The diamond drill core was transported from the drill site to the Boeuf Mort camp where all geologic logging and sampling was conducted. Sample intervals were marked in advance by the Project geologists. The saprolite core was halved with a knife, while fresh rock core was sawn with a powered diamond saw. The original assay lengths range from 0.1 to 4.3 m with an average of 1.0 m. A total of 10,693 samples were taken. The presence of dispersed zones of very narrow sulphide bands, in some cases, forced sample intervals that did not always conform to the actual lithologic breaks. The sawn half-core was bagged, labelled on site, and sent out for assaying.

Sample bags were routinely placed in plastic rice bags and sealed to prevent tampering between the campsite and the laboratory. The remaining half core was returned to the core box and stored for future reference.

Rock quality description (RQD) measurements were completed on selected intervals in seven drillholes during the 1998 campaign. Magnetic susceptibility measurements were completed for 18 drillholes during this campaign (MO9601 to MO9618).

Bulk density measurements on drill core were not performed on a regular basis. The densities used for previous resource estimations utilized bulk densities taken from equivalent or nominal rock types (not described).

The diamond core and channel samples collected in the Montagne d'Or prospect area during the 1996/1998 drilling campaign were dispatched to six separate laboratories for sample size reduction, homogenization, and assay determination. Analytical methodologies utilized were typically fire assayed with an atomic absorption finish. A few samples were assayed by fire assay (FA) with a gravimetric finish. These are appropriate and standard methodologies for gold analysis. There is no documentation in the Project files related to the certification of any of the laboratories used to analyze the Montagne d'Or prospect samples. It was not industry standard of the time to undergo certification procedures.

The QA/QC procedures for the Montagne d'Or Prospect analytical work prior to 1998 utilized check assays performed on quarter core, the remaining half of re-sawn split half core. Most quarter-core samples were collected from barren core (<0.05 g/t Au) and used for blank material. Since the samples were not extracted from the same pulp, the samples are more correctly termed field duplicates. No data are available for assay standards included with any of the drill or channel sample analyses. Internal check assay information is provided for five of the six laboratories that were used for gold assaying.

RSG (2004) provided a review of the QA/QC results obtained during the history of the drilling and they concluded the following: the results of the RSG Global statistical assessment of the quality control data suggest that the SGS Cayenne and CanTech laboratories were producing assay results of an acceptable precision and unknown accuracy, but that the SGS France and Cone Colorado laboratories were not producing assays of an acceptable precision. The various coarse reject check assaying programs indicate that there are serious problem at all or some of the laboratories and that precision levels from all the check assay programs are unacceptable. Correlation between assay pairs is very poor with significant bias shown in some instances. The accuracy of the data produced by each laboratory cannot be assessed without standard reference assay data, and this is a material flaw in the check assay programs completed to that date.

In 2007, Golden Star conducted a modern QA/QC analysis during a re-assay program of the historical drill core at the Paul Isnard deposit. This consisted of re-sampling of the core from a wide distribution of drillholes, insertion of blanks and standards, and submitting all these to an accredited laboratory.

The laboratory employed industry standard sample preparation and the techniques of analyses were appropriate for the level of gold mineralization. The results of the QA/QC verified the credibility of the 2007 re-assay results. This is discussed further in Section 11.3.

11.2 Columbus Drill Program

The following description of sample preparation and core handling protocols applies to all drilling carried out by Columbus to date on the Montagne d'Or prospect. The next sections describe the 2011 and 2012 logging and sampling procedures, which were upgraded during the subsequent programs (geotechnical logging, core photography, air transport to Cayenne, use of Geotic software, assays on 50 g split by fire assay with atomic adsorption finish (FA/AA), assays above 5 g/t Au re-assayed by gravimetrics, refer also Section 12). Program details on the current logging, sampling and QA/QC protocols were discussed in detail with Columbus staff during the site visits by SRK and their systematic application with respect to the Project was confirmed.

11.2.1 Core Logging and Sampling

Drill core is placed in plastic trays at the drill site by the drill crew. Drillers either transport the core to the end of the road for pickup by Columbus personnel or directly to the core shack in the Citron Camp.

Once in the camp the core boxes are opened and placed in order on logging racks within the core logging facility (Figure 11-1). If space is not available then the core is stored in core racks adjacent to the logging facility.

Source: Columbus, 2015

Figure 11-1: Core Logging Facility at the Citron Exploration Camp

The drill core is washed to remove any dirt or grease and reconstituted. The core is measured to ensure that there are markers every metre. Basic geotechnical logging is initially undertaken, measuring recovery and RQD.

The core is descriptively logged and marked for sampling by Columbus geologists. Logging and sampling information is entered into a computer using Excel software. Selected intervals of core are photographed however the entire drillhole is not systematically photographed.

After logging the core is prepared for sampling. A line is drawn down the core and the cutter uses this as a guide. The entire drillhole is then cut. A Columbus geologist does the actual sampling.

The core is sampled at 1 m intervals using the measuring blocks prepared upon initial receipt of the drill core as a guide. The entire drillhole is sampled at an average of 1 m intervals; sample lengths are adjusted to honor lithological contacts and mineralized intervals. Half of the drill core is placed in a plastic sample bag while the other half is retained din the core box for future reference. Saprolite material is cut with a knife and half placed in a textile bag for assay and the other half returned to the core box The samples and sample bags are numbered sequentially in advance allowing for the insertion standard reference samples, duplicates and blanks. The plastic sample bags are placed in larger rice bags and sealed for shipping. The sample bags are then sent by air transport to Cayenne and dropped off by SOTRAPMAG personnel to the Filab depot in Cayenne, followed by road transport from Cayenne to the laboratory in Paramaribo, Suriname for preparation and analyses.

All the core from Columbus's drilling is stored in covered core racks at the Citron exploration camp (Figure 11-2).

Source: Columbus, 2015

Figure 11-2: Core Racks at the Citron Exploration Camp

11.2.2 Density Measurements

Columbus measures the bulk density of representative samples of the various rock types and not the bulk density in each drillhole. They used a conventional bulk density scale with a basket suspended below the scale to allow immersion in water. Samples are not coated in paraffin wax, however, the core was observed to be generally solid with very little pores. Saprolite was wrapped in cellophane.

The following measurement methodology was employed:

- Weigh the sample to determine the dry mass;
- Place the sample in a basket and weigh it, suspended from a balance, in (under) water. Subtract the weight of the basket in (under) water, to determine the mass of the sample in water; and
- The relative dry bulk density, a unit-less ratio, is calculated as the dry mass of the sample in air divided by the difference in the mass of the sample in air and the mass of the sample in water.

The scale is zeroed out before each use and the weight of the basket holding the core is repeatedly measured.

Water density is assumed to be 1.0 t/m^3 with no adjustment made for changes in water temperature. Since all measurements were performed indoor in normal air temperatures, the actual water density should range between 0.999 t/m^3 at 15° C to 0.997 t/m^3 at 25°C. Therefore, assuming a value of 1.0 t/m^3 for water density will not introduce a significant bias in the estimate and is to industry standards.

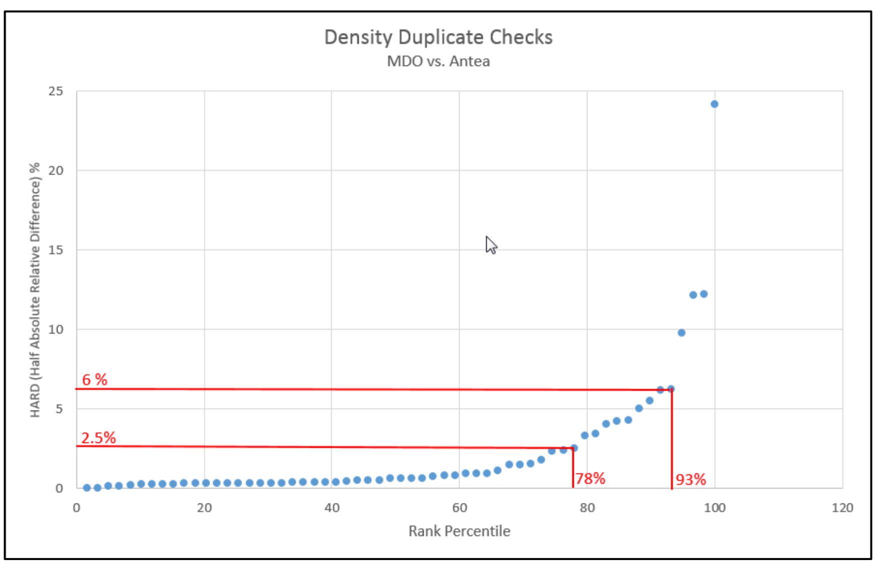
As of November 2014, Columbus had made a total of 3,323 bulk density measurements on Montagne d'Or drill core. Bulk Density measurements were recorded for 9 different rock units (Table 11-1).

Table 11-1: Listing of Montagne d'Or Prospect Dry Rock Density Measurements

Rock Type	Number of Measurements	Minimum	Maximum	Average Density g/cm^3
Saprolite	412	1.17	2.67	1.736
Saprolite-Rock Transition	175	1.61	2.89	2.412
Felsic Tuff	1,319	1.55	4.53	2.893
Mafic Volcanics	442	2.73	4.33	3.131
Granodiorite	626	2.58	3.20	2.749
Feldspar Porphyry	73	2.62	2.87	2.777
Quartz-Feldspar Porphyry	117	2.66	3.10	2.779
Lapilli Tuff	80	2.63	3.17	2.827
Diabase Dikes	363	2.69	3.16	3.004
Total	**3607**			

Source: SRK, 2016

Columbus also conducted independent density checks on 59 samples from the six most significant lithologies. The duplicate density checks were completed by an Antea Laboratory in Orleans France. Gabarit Geotic is an affiliate of the international geotechnical firm Antea. The results of the duplicate checks were analyzed with a half absolute relative difference (HARD) plot as shown in Figure 11-3. The plot shows very good correlation between the density check pairs with 93% of the pairs showing HARD values below 6% and 78% of the pairs below 2.5% HARD. The density check results confirm that the density measurements obtained by Columbus personnel have reliable precision to support the resource estimation and mine planning of the current study.



Source: SRK, 2016

Figure 11-3: HARD Plot of Density Duplicates

11.2.3 Sample Preparation and Analysis

Columbus staff log and sample drill core but do not carry out any form of sample preparation (crushing/pulverizing) or analytical work on Project samples. All Project analytical work including sample preparation and analytical work is completed by FILAB at their laboratory in Paramaribo, Surinam.

FILAB established for several years a system of Quality Management and Safety to meet customer requirements (standards ISO17025 and ISO9001). FILAB is accredited by the DKD (now the DakkS) and the SAFRAN Group and approved by DF control PMUC. The following description is sourced from documentation provided by FILAB.

After samples are received at the laboratory, then weighed and dried in furnaces at a temperature <130°C. They are then crushed and ground to a 70% <2.5 mm. From this grind a 300 to 400 g split is pulverized to 90% <100 µm. All equipment is cleaned by air after the processing of each sample.

Gold concentrations for the Columbus program are analyzed by FILAB using a 30 g sample split and fire assay pre-concentration methods followed by an atomic absorption spectroscopy finish (FA/AAS). The detection limit for this method is 0.01 ppm Au.

Gravimetric analysis was conducted on samples above a 5 g/t Au value for the 2013 and 2014 drilling program (the threshold is not reported for the earlier drilling and cannot be verified as personnel involved is no longer on site) and results from the gravimetric analysis were prioritized over FA in the database.

Induced coupled plasma (ICP) analysis for up to 40 elements but routinely only for Silver (Ag) and Copper (Cu) are done using Aqua Regia digestion on a 0.25 g subsample.

FILAB routinely inserts blanks and certified reference materials (standards) into each batch of samples as an internal check.

11.3 Quality Assurance/Quality Control (QA/QC)

The QA/QC of all exploration data prior to October, 2015 has been presented in prior technical reports. The information presented below relates to the most recent exploration drilling conducted by Columbus during 2015 and 2016.

The Columbus QA/QC protocol of the 2015-2016 drilling programs included six different commercially certified standard reference materials for Au, blanks and field duplicates. The Columbus standards ranged between 0.599 to 8.671 g/t Au, which represents the typical levels of gold mineralization in the deposit. Standards are blindly inserted to the sample stream at a rate of 1:20 samples. The results of the standard analysis must be within ±2 standard deviations of the mean to pass the initial validation. In the case of standard result is between ±2 and ±3 standard deviations, a more complete check is made to determine if the result is valid or not. If the standard is outside a mineralized zone, reanalysis of the batch is not necessary. If two standards in succession, return results between ±2 and ±3 standard deviation, the batch is typically reanalyzed. If the standard value is outside ±3 standard deviations, the value is considered as erroneous and the entire batch is reanalyzed by the laboratory.

Columbus blanks are blindly inserted with at least one per batch with the blank located after an interpreted zone of mineralization. Blanks used during the program came from a granite quarry located near Cayenne. The blank analysis is considered valid if its value is lower than 5 times the limit of detection (0.005x5 = 0.025 ppm), confirming that no contamination occurred. If the analysis is beyond 5 times the limit of detection, the entire batch is reanalyzed by the laboratory.

The insertion of field duplicate pairs allows checking for problems that might affect the reproducibility (precision) of assay results that could potentially arise from the nugget effect. In the case of RC drilling, reproducibility might also be affected poor homogenization of the sample when passed through the field splitter.

The laboratory conducts four types of internal QA/QC. They utilize two types of duplicates, standards and blanks. The laboratory uses duplicate pulps, generated and analyzed at a typical rate of 1:30 samples. Duplicate analyses of the same pulp are run at a typical rate of 1:15 samples.

QA/QC results are compiled in Excel as monthly reports. A representative set of standards at three typical grades and the blank results from the 2015-2016 drilling program are presented in Figure 11-4 to Figure 11-7.

The results of field duplicate analysis pairs are assessed for the relative error by way of the HARD statistical method described by Stanley & Lawey (2007) and currently used as an industry best practice statistical analysis for assay precision/reproducibility. The HARD values were calculated using:

$$\text{HARD} = [|x1 - x2| / (x1 + x2)] / 2$$

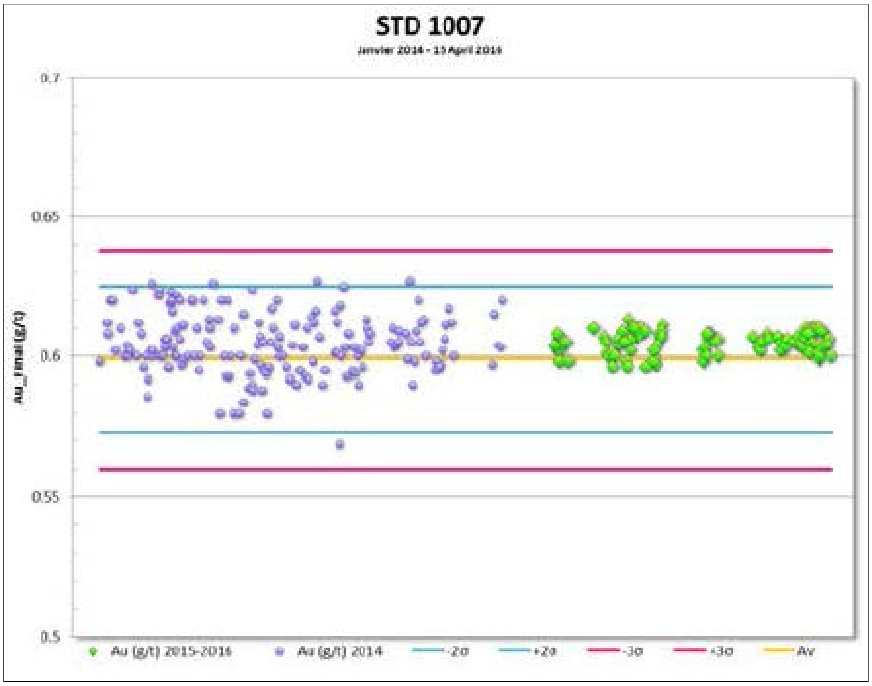
Where x1 and x2 are duplicate pair results. HARD values for a population of duplicates are typically plotted by rank percentile. They are presented for Phase III diamond core samples in Figure 11-8 and for Phase III RC samples in Figure 11-9. The HARD statistics are also summarized for the Phase III campaign in Table 11-2. For this analysis the results were broken down into two bins based on assay values for the original samples, one for assay results >= 10X the laboratory analytical limit (i.e., 0.05 ppm) and < 1 g/t, the other for assays results >= 1 g/t. This was done to validate any potential effect of lower and higher grades on the precision of the results. Samples with grades below 0.05 were not included in the analysis as it is thought they would introduce excessive natural uncertainty into the statistical results.

Duplicate analyses were also completed on pulp materials at an outside check laboratory. A total of 891 pulps prepared at Filab were sent to SGS in Lima Peru for independent analysis. A scatter plot of all data confirms there is no systematic bias at either lab as also shown by the SRM results at Filab. The pulp duplicate data was analyzed in a similar manor to the field duplicates. The data was sorted into tow grade ranges and then plotted on HARD plots. The results are shown in Figure 11-10 and Figure 11-11. The lower grade data, less than 1 g/t Au, shows that 82% of the data pairs have less than 25 % HARD and that 50% of the data pairs have a HARD value below 10%. In the higher grade data, Au above 1 g/t, 73% of the data pairs have a HARD value less than 25% and 42% of the data pairs have a HARD value below 10%. Overall the pulp duplicate results are very good showing reliable precision at the primary laboratory.

Table 11-2: HARD Statistics for 2015-2016 Field Duplicates

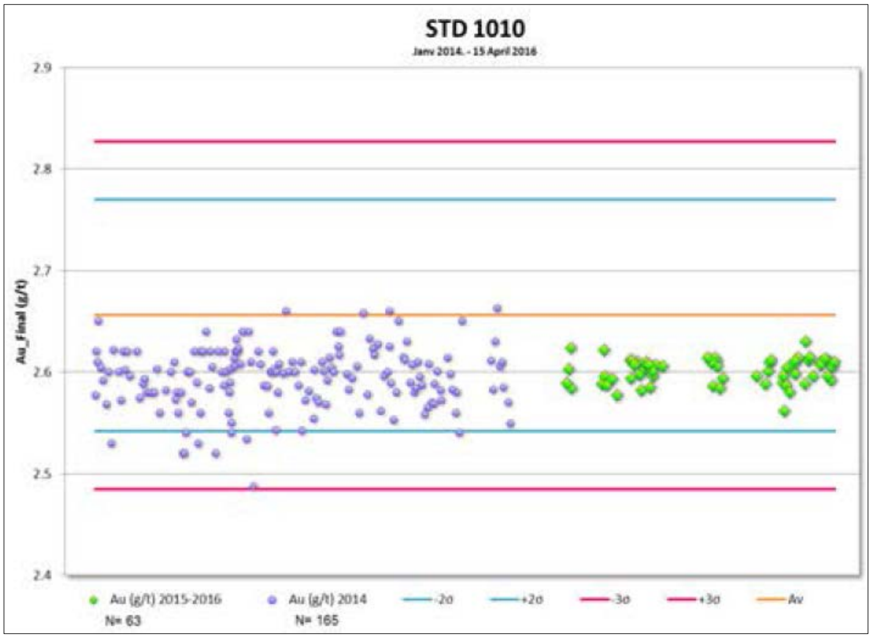
Sample Type/Grade Range	Quantity	HARD < 25% (#)	HARD < 25% (%)
RC / < 1.0 g/t Au	129	72	56
RC / >= 1.0 g/t Au	39	25	64
Core / < 1.0 g/t Au	73	47	64
Core / >= 1.0 g/t Au	21	9	43

Source: Columbus 2016



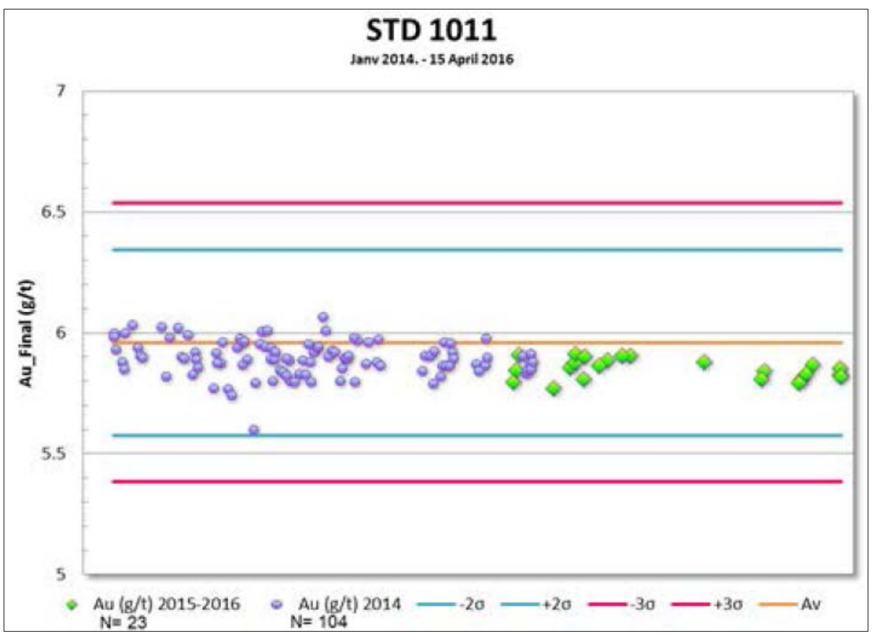
Source: Columbus, 2016

Figure 11-4: Results of Au Standard at 0.599 g/t



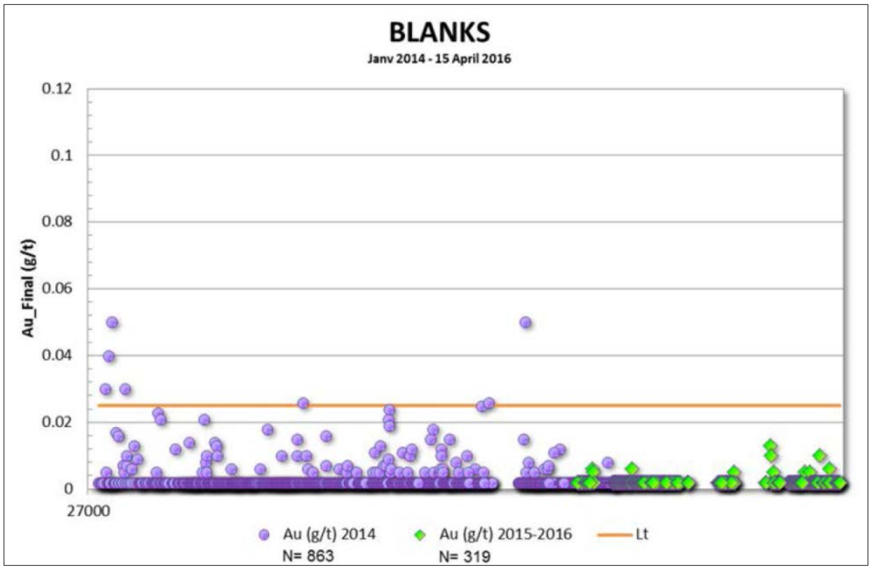
Source: Columbus, 2016

Figure 11-5: Results of Au Standard at 1.807 g/t



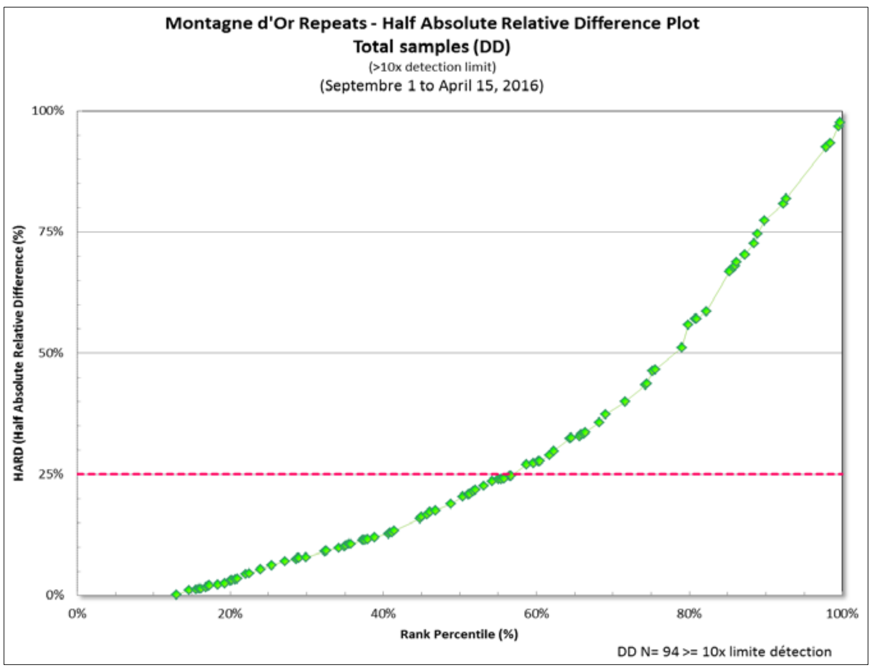
Source: Columbus, 2016

Figure 11-6: Results of Au Standard at 5.96 g/t



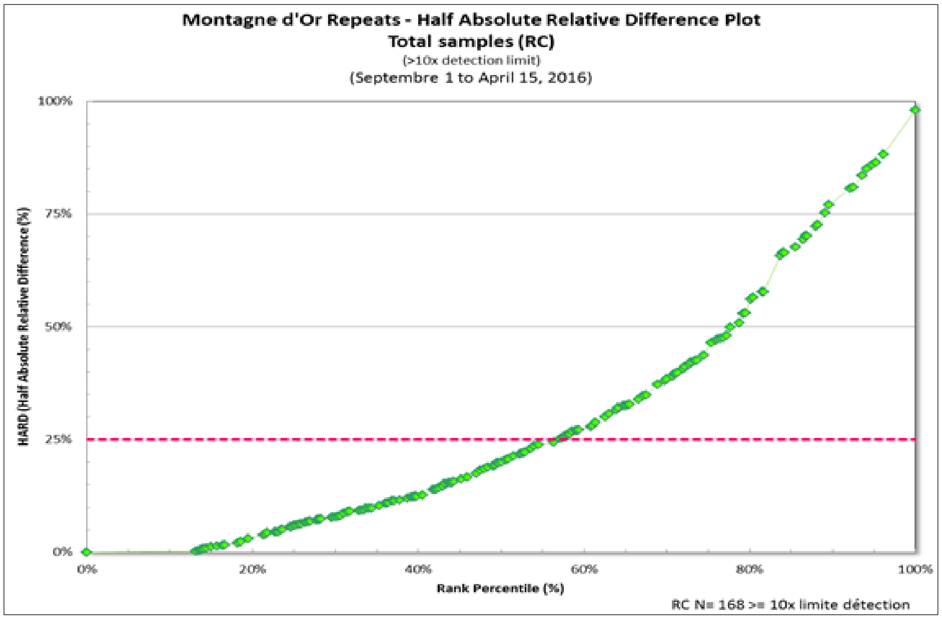
Source: Columbus, 2016

Figure 11-7: Results of all Blank Analyses



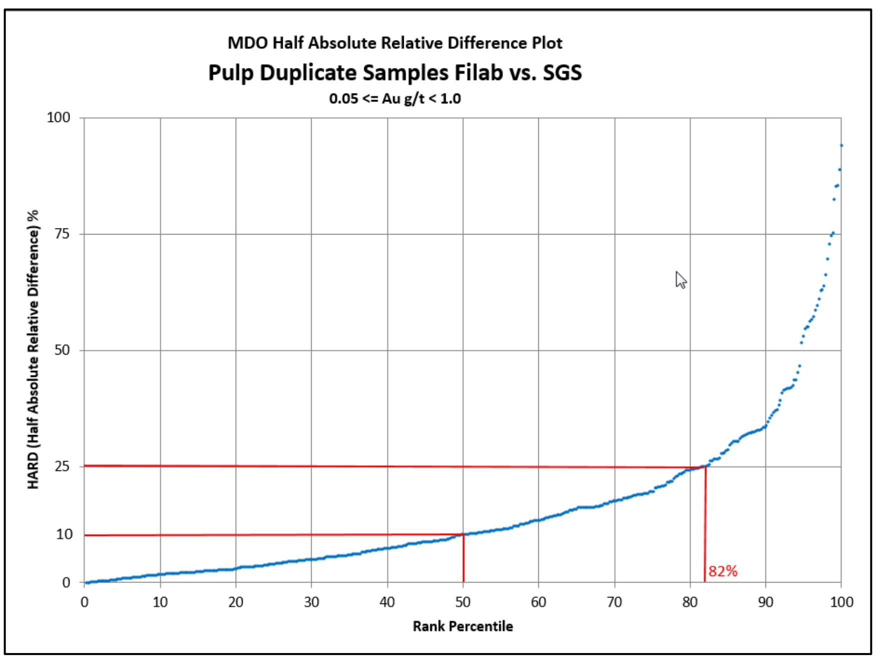
Source: Columbus, 2016

Figure 11-8: HARD Plot of all Core Field Duplicates



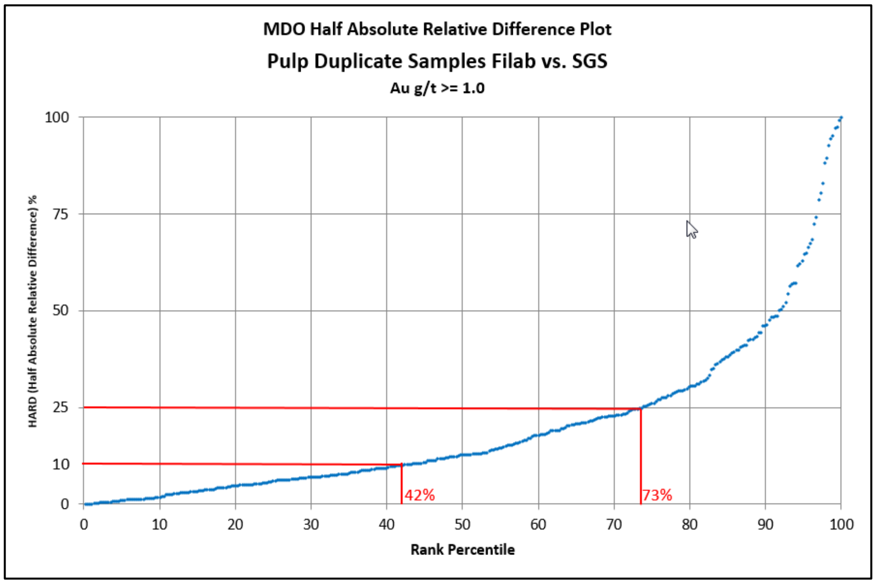
Source: Columbus, 2016

Figure 11-9: HARD Plot of all RC Field Duplicates



Source: SRK, 2016

Figure 11-10: HARD Plot of Lower Grade Pulp Duplicates



Source: SRK, 2016

Figure 11-11: HARD Plot of Higher Grade Pulp Duplicates

11.3.1 Conclusions

SRK is of the opinion that best professional judgment, and appropriate exploration and scientific methods were utilized in the preparation and analysis of the samples used in this report. SRK has reviewed the QA/QC results of the 2015-2016 drilling programs. SRK finds that the QA/QC program was well planned, executed and monitored. The standards are all certified and of appropriate levels of Au mineralization. The blank material is sufficiently hard so that it will scrub the sample preparation equipment to reveal any cross contamination. The results of the standards confirm there is no bias of the analytical lab. They also confirm that the laboratory has produced results with industry standard precision and accuracy. The blanks submitted with the QA/QC samples have shown that cross contamination or possible sample mix-ups are rare and do have a material impact on the analytical results. The field duplicates show that sample collection, labeling and preparation were conducted at a high level of care and quality. The pulp duplicates show reliable precision of the primary laboratory.

12 Data Verification

12.1 Procedures

The database constructed prior to April 1, 2016 has been validated and reported in previous Technical Reports in order to support their resource estimations. SRK validated the 2015-2016 sample assay database by conducting systematic comparisons between the original assay certificate PDF copies to the electronic excel spreadsheet. Systematically spaced data every 20^{th} entry was checked from a range of certificates that cover all of the new assays. A total of 504 entries were checked, representing 6.0% of the new assay data. No discrepancies were found.

12.2 Limitations

SRK was not materially limited in its access to the supporting data used for the resource estimation. The database verification is limited to the procedures described above. All Mineral Resource data relies on the industry professionalism and integrity of those who collected and handled it. SRK is of the opinion that appropriate scientific methods and best professional judgment were utilized in the collection and interpretation of the data used in this report. However, users of this report are cautioned that the evaluation methods employed herein are subject to inherent uncertainties.

12.3 Opinion on Data Adequacy

It is SRK's opinion that the drillhole data is adequate to support the resource estimation of this report at the current level of resource classification. The database was constructed by Columbus under industry standard QA/QC protocols. Columbus maintains the database using GeoTic IOG an integrated database management system specifically designed to minimize the possibilities for data entry or data transfer errors. SRK's evaluation and subsequent validation of the database has provided good confidence in the data files.

13 Mineral Processing and Metallurgical Testing

The metallurgical program for the Project was based on earlier metallurgical studies that were conducted as part of a PEA of the Project during 2014 and 2015 by BV and documented in their report, "Metallurgical Testing to Recover Gold and Silver on Samples From the Montagne d'Or Project, French Guiana, April 6, 2015." The PEA metallurgical program evaluated three process options, including whole-ore cyanidation, a combination of gravity concentration followed by cyanidation of gravity tailing, and gravity concentration followed by gold flotation from the gravity tailing and cyanidation of the flotation concentrate. Based on the results of the PEA, the BFS metallurgical program focused on the development of a process flowsheet that included gravity concentration followed by cyanidation of the gravity tailings and intensive cyanide leaching of the gravity concentrate. This program was conducted by several different commercial laboratories including: BV, Pocock Industrial, ALSy, SGS Canada, and FLSmidth and the results of these studies are fully documented in the following reports:

- "Bankable Feasibility Metallurgical Testing of Samples from the Montagne d'Or Gold Project, French Guiana"; BV, June 3, 2016.
- "Comminution Testing – Montagne d'Or Project"; ALS, February 5, 2016.
- "Recovery of Gold from Montagne d'Or Samples"; SGS Canada Inc., July 20, 2016.
- "Flocculant Screening, Sedimentation and Pulp Rheology Studies Conducted for the Montagne D'OR Project"; Pocock Industrial Inc., April 2016.
- EGRG Gravity Test Work Report, Montagne d'Or Project in French Guiana"; FLSmidth Ltd – Knelson Technologies, April 8, 2016.
- Gravity Circuit Modeling Report, Nordgold, Montagne' d'Or Project; FLSmidth Ltd, April 18, 2016.

13.1 Test Composite Characterization

The metallurgical program was conducted on three master composites, 15 variability composites representing different ore lithologies and grade ranges, and seven variability composites representing seven mining phases that were identified at the start of the program.

13.1.1 Master Composites

The three master composites were developed to represent the upper felsic zone (UFZ), lower favourable zone (LFZ), and saprolite/saprock. The UFZ and LFZ master composites were formulated from whole-core drillhole intervals derived from six metallurgical drillholes, which were planned based on the following criteria:

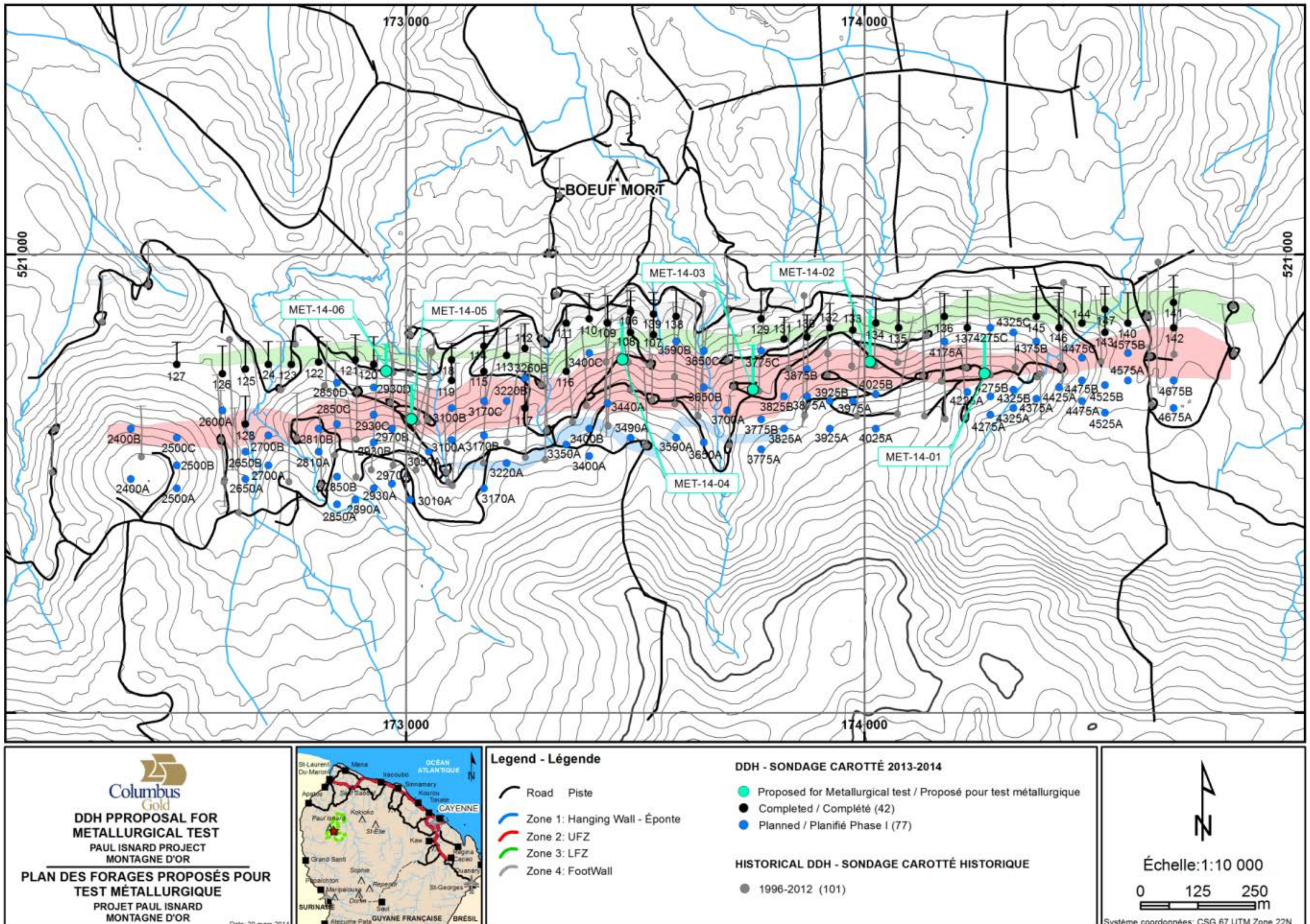
- Twinning of previous drillholes that intersected representative gold-copper intersections of variable grades across the principal felsic volcanic hosted UFZ and mixed volcanic hosted LFZ.
- A minimum of four intersections across UFZ and two across the LFZ, uniformly distributed along the east-west strike extent of the Montagne d'Or deposit.
- Intersections of the UFZ and LFZ in fresh rock below the weathered and oxidized saprolitic layer.

The drillhole locations and core intervals selected to formulate the UFZ and LFZ master composites are shown in Table 13-1 and Figure 13-1.

Table 13-1: Drillholes and Intervals used for the UFZ and LFZ Master Composites

Hole ID	Zone	From (m)	To (m)	Length (m)	Core Wt. (kg)
MET-14-01	UFZ	108.0	145.0	37	315
MET-14-02	UFZ	37.8	54.6	17	143
	UFZ	68.6	71.6	3	26
	UFZ	77.6	80.6	3	26
	UFZ	89.6	94.6	5	43
	LFZ	163.1	190.2	27	230
			Total	**55**	**467**
MET-14-03	UFZ	37.0	163.0	126	1,071
MET-14-04	UFZ	51.8	78.8	27	230
	LFZ	125.0	150.6	26	218
			Total	**102**	**870**
MET-14-05	UFZ	71.0	109.0	38	323
	UFZ	130.0	136.0	6	51
			Total	**44**	**374**
MET-14-06	LFZ	79.6	103.5	24	203
			Total Core	**388**	**3,300**

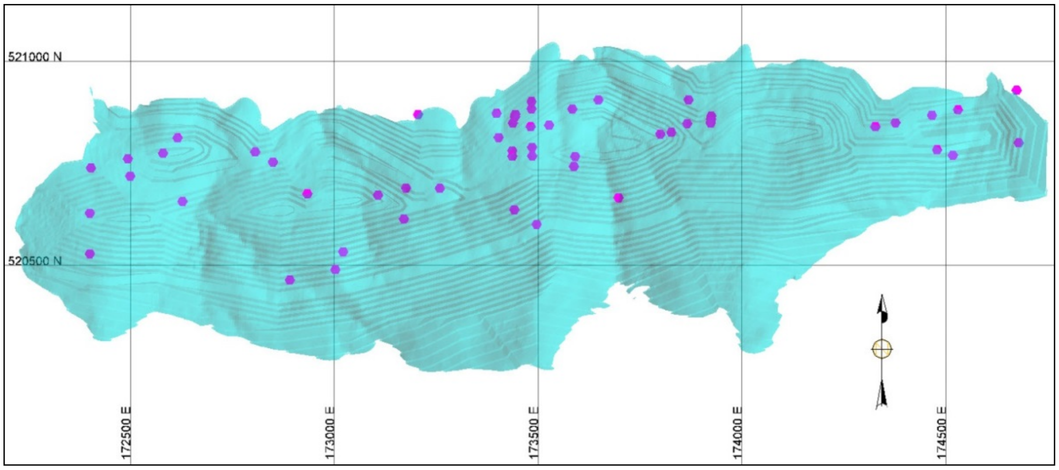
Source: Nordgold, 2015



Source: Nordgold 2015

Figure 13-1: Drillhole Locations for the UFZ and LFZ Master Composites

The saprolite/saprock master composite was formulated from two lithologies, saprolite and saprock at a 75% and 25% contribution, respectively. A total of 56 samples averaging 1 m in length provided approximately 324 kg of material. The location of the selected drillholes is shown in Figure 13-2.



Source: SRK 2016

Figure 13-2: Saprolite/Saprock Master Composite Drillhole Locations

13.1.2 Lithology Variability Composites

The lithology variability composites were developed for each of the five major rock types in three grade ranges. The composite lithologies include; saprolite, saprolite-rock, felsic tuff, granodiorite, and mafic volcanics. Three relative grade ranges were defined as "lower grade" composed of samples with Au assays in the range of 0.7 to 1.5 g/t Au, "average grade" composed of samples with Au assays in the range of 1.51 to 2.25 g/t Au and "higher grade" composed of samples with Au assays in the range of 2.25 to 3.1 g/t Au. Each composite sample was comprised of approximately 16 to 23 samples averaging 1 m in length which accumulate to approximately 60 kg of material. Table 13-2 lists the predicted average grades and approximate weights for each composite.

Table 13-2: Summary of Selected Lithology Variability Composites and Grade Ranges

Grade Range	Lithology	Average Au (g/t)	Average Cu (%)	Total Length of Core Sampled (m)	Approximate Mass (Kg)
Lower	Saprolite	1.09	0.02	22.8	61
	Saprock	1.04	0.03	23.0	78
	Felsic Tuff	1.05	0.10	23.2	60
	Granodiorite	1.08	0.06	23.1	60
	Mafic Volcanic	1.10	0.06	22.2	58
Average	Saprolite	1.70	0.03	16.9	45
	Saprock	1.81	0.03	20.8	68
	Felsic Tuff	1.84	0.12	23.0	60
	Granodiorite	1.91	0.01	21.3	55
	Mafic Volcanic	1.84	0.16	23.0	60
Higher	Saprolite	2.50	0.02	2.2	6
	Saprock	2.69	0.50	8.4	32
	Felsic Tuff	2.62	0.18	23.1	60
	Granodiorite	2.61	0.13	16.2	42
	Mafic Volcanic	2.59	0.13	23.0	60

Source: SRK 2016

13.1.3 Mining Phase Variability Composites

Seven variability composites were prepared to reflect the weighted contribution of each ore lithology that would be mined during each of the seven mining phases that were defined in the 2015 PEA. Table 13-3 shows the contribution from each lithology during each phase of mining and was the basis for creation of each of the mining phase variability composites.

Table 13-3: Mining Phase Variability Composites and Lithological Contribution

Mining Phase	Lithology	Percent of Lithology Mined During Phase
Phase 1	Saprolite	18
	Saprock	5
	Felsic Tuff	70
	Granodiorite	7
	Mafic Volcanic	0
	All	100
Phase 2	Saprolite	34
	Saprock	11
	Felsic Tuff	53
	Granodiorite	2
	Mafic Volcanic	0
	All	100
Phase 3	Saprolite	15
	Saprock	7
	Felsic Tuff	68
	Granodiorite	1
	Mafic Volcanic	9
	All	100
Phase 4	Saprolite	6
	Saprock	1
	Felsic Tuff	81
	Granodiorite	12
	Mafic Volcanic	0
	All	100
Phase 5	Saprolite	8
	Saprock	2
	Felsic Tuff	77
	Granodiorite	13
	Mafic Volcanic	0
	All	100
Phase 6	Saprolite	3
	Saprock	1
	Felsic Tuff	85
	Granodiorite	4
	Mafic Volcanic	7
	All	100
Phase 7	Saprolite	3
	Saprock	1
	Felsic Tuff	52
	Granodiorite	0
	Mafic Volcanic	44

Source: SRK, 2016

13.1.4 Composite Head Analyses

Gold analyses were performed by standard fire-assay in triplicate along with metallic screen analyses on each master composite and variability composite and the results are presented in Table 13-4. The UFZ and LFZ master composites averaged 2.18 and 1.61 g/t Au, respectively. The saprolite/saprock master composite averaged 1.76 g/t Au. The variability composites ranged from 0.95 to 3.79 g/t Au and averaged 1.87 g/t Au.

Table 13-4: Gold Analyses on the Master and Variability Composites

Composite ID		By Direct Fire-assays (Au g/t) Cut A	Cut B	Cut C	Cut D	Cut E	Cut F	Metallic Au (g/t)	Average Au (g/t)
Master Comp.	UFZ Master Comp.	1.36	1.04	2.36	4.07			2.07	2.18
	LFZ Master Comp.	3.21	1.01	0.89				1.32	1.61
	Saprolite/Saprock Master Comp.	1.69	1.21	1.18	1.16	1.50	7.39	1.61	1.76
		2.07	1.24	1.30	1.64	1.27	1.28		
		1.28	1.60	1.07	2.42	1.19	1.26		
Average of Master Composites									**1.85**
Mining Phase Variability Comp.	Mining Phase 1 Variability Comp.	1.27	1.54	2.70	1.56			1.95	1.80
	Mining Phase 2 Variability Comp.	1.26	4.85	4.84				4.37	3.83
	Mining Phase 3 Variability Comp.	0.98	1.44	1.19				1.29	1.23
	Mining Phase 4 Variability Comp.	1.74	1.49	1.42				2.67	1.83
	Mining Phase 5 Variability Comp.	1.35	2.40	2.24				2.69	2.17
	Mining Phase 6 Variability Comp.	0.85	1.52	1.11				1.35	1.21
	Mining Phase 7 Variability Comp.	2.35	1.65	1.42	1.52			1.52	1.69
Lithology Variability Comp.	Granodiorite Average Grade Variability Comp.	1.30	9.94	2.73				1.18	3.79
	Granodiorite Higher Grade Variability Comp.	1.72	2.19	1.50				3.56	2.24
	Granodiorite Lower Grade Variability Comp.	1.18	0.83	1.03				0.76	0.95
	Saprolite Average Grade Variability Comp.	1.77	0.98	1.68				1.93	1.59
	Saprolite Higher Grade Variability Comp.	1.03	0.91	1.78				0.97	1.17
	Saprolite Lower Grade Variability Comp.	2.28	0.50	0.91				1.05	1.18
	Saprock Average Grade Variability Comp.	0.91	1.91	1.09	0.90			1.34	1.23
	Saprock Higher Grade Variability Comp.	2.65	2.09	2.20				2.51	2.36
	Saprock Lower Grade Variability Comp.	0.72	1.15	1.01				1.36	1.06
	Felsic Tuff Average Grade Variability Comp.	3.58	2.16	1.65				1.58	2.24
	Felsic Tuff Higher Grade Variability Comp.	2.20	1.87	2.55				2.74	2.34
	Felsic Tuff Lower Grade Variability Comp.	1.12	0.66	0.73				1.29	0.95
	Mafic Volcanics Average Grade Variability Comp	1.50	2.28	2.40	2.38			2.24	2.16
	Mafic Volcanics Higher Grade Variability Comp	2.48	2.50	3.67				2.77	2.86
	Mafic Volcanics Lower Grade Variability Comp	1.23	1.21	1.15				1.44	1.26
Average of Variability Composites									**1.87**
Overall Average									**1.85**

Source: BV, 2016

13.2 Comminution Studies

A comminution test program was conducted by ALS on comminution composites representing the UFZ and LFZ master composites and six lithologies representing the major rock types which included, saprolite, saprolite-rock, felsic tuff, granodiorite, mafic volcanics, and mineralized diabase. The comminution test program included:

- Bond low energy impact and abrasion testing on each of the eight comminution composites;
- Bond ball mill work index determinations;
- SAG Mill Comminution (SMC) testing; and
- SAG Power Index (SPI) testing (conducted and reported by SGS).

13.2.1 Low Energy Impact and Abrasion Test Work

Bond low energy impact tests (Cwi) were completed on six of the comminution composites, but could not be completed on the saprolite and saprock composites due to the fine size of these two composites. It was found that the Cwi for the ore lithologies ranged from 5.73 to 7.49 kWh/t and averaged 6.39 kWh/t. The diabase composite, which represents the waste rock and may be contributed to the plant feed as dilution, was found to be significantly harder at 9.94 kWh/t.

13.2.2 Bond Abrasion Index Test Work

Bond abrasion index tests (Ai) were completed on all comminution composites except the saprolite composite. An average abrasion index value of 0.06 was recorded. Ai values less than 0.20 are considered only mildly abrasive.

13.2.3 Bond Ball Mill Work Index

Bond ball mill work index (Bwi) tests were conducted using a closing screen of 106 µm on the eight comminution composites and 15 variability composites and the results are summarized in Table 13-5. The Bwi for these composites ranged from 6.0 to 16.6 kWh/t and averaged about 11.7 kWh/t.

Table 13-5: Bond Ball Mill Work Index Test Results

Composite ID	Bwi
	kW-hr/t
Granodiorite Average Grade Variability Composite	12.2
Granodiorite Higher Grade Variability Composite	12.3
Granodiorite Lower Grade Variability Composite	12.5
Sap-Rock Higher Grade Variability Composite	6.9
Sap-Rock Lower Grade Variability Composite	6.0
Sap-Rock Average Grade Variability Composite	6.7
Felsic Tuff Average Grade Variability Composite	11.7
Felsic Tuff Higher Grade Variability Composite	11.7
Felsic Tuff Lower Grade Variability Composite	11.7
Mafic Volcanics Average Grade Variability Comp.	15.6
Mafic Volcanics Higher Grade Variability Composite	16.6
Mafic Volcanics Lower Grade Variability Composite	16.5
Saprolite Comminution Composite	
Saprock Comminution Composite	7.3
Felsic Tuff Comminution Composite	11.0
Granodiorite Comminution Composite	12.4
Mafic Volcanics Comminution Composite	14.4
Diabase Comminution Composite	13.4
UFZ Comminution Composite	11.3
LFZ Comminution Composite	12.1

Source: ALS, 2016

13.2.4 SMC Test Work

SMC tests were performed on all composites except the saprolite composites. The results of these tests are summarized in Table 13-6. The A*b parameter derived from the SMC tests, which indicates the relative resistance to breakage in a SAG mill, ranged from 25 for the mafic volcanics variability composites to 176 for the saprock variability composite. It should be noted that smaller A*b numbers indicate harder ores and larger numbers indicate softer ores.

Table 13-6: SMC Test Results

Sample ID	Dwi KWh/m^3	A	b	t_a	A x b	SCSE kWh/t
Granodiorite Average Grade Variability Composite	7.07	57.6	0.69	0.37	39.7	10.1
Granodiorite Higher Grade Variability Composite	7.74	69.6	0.51	0.33	35.5	10.6
Granodiorite Lower Grade Variability Composite	6.96	55.5	0.72	0.37	40.0	10.0
Sap-Rock Higher Grade Variability Composite	1.84	72.3	1.97	1.41	142	6.1
Sap-Rock Lower Grade Variability Composite	1.56	74.5	2.18	1.66	162	6.0
Sap-Rock Average Grade Variability Composite	1.64	76.5	2.01	1.58	154	6.0
Felsic Tuff Average Grade Variability Composite	6.36	56.2	0.76	0.41	42.7	9.6
Felsic Tuff Higher Grade Variability Composite	6.54	59.1	0.73	0.40	43.1	9.8
Felsic Tuff Lower Grade Variability Composite	8.26	62.6	0.54	0.31	33.8	10.9
Mafic Volcanics Average Grade Variability Comp	12.4	67.5	0.37	0.21	25.0	13.7
Mafic Volcanics Higher Grade Variability Comp.	13.1	84.4	0.28	0.20	23.6	14.1
Mafic Volcanics Lower Grade Variability Comp.	11.2	65.3	0.41	0.23	26.8	13.0
Saprolite Comminution Composite						
Saprock Comminution Composite	1.45	71.7	2.46	1.78	176	5.8
Felsic Tuff Comminution Composite	7.19	53.9	0.74	0.36	39.9	10.3
Granodiorite Comminution Composite	7.26	61.0	0.63	0.36	38.4	10.3
Mafic Volcanics Comminution Composite	9.66	45.9	0.72	0.27	33.0	12.0
Diabase Comminution Composite	12.9	85.8	0.27	0.20	23.2	14.2
UFZ Comminution Composite	7.41	61.0	0.62	0.35	37.8	10.4
LFZ Comminution Composite	7.06	57.5	0.69	0.37	39.7	10.2

Source: ALS, 2016

13.2.5 SPI Test Work

SAG Power Index (SPI) tests were conducted by SGS on the eight comminution composites in order to confirm the findings from the SMC tests. The SPI test, expressed in minutes, is defined as the time necessary to reduce an ore sample from 80% passing 12.5 mm to 80% passing 1.7 mm, and is a measure of the hardness of the ore from the perspective of SAG mill grinding. The results of these tests are summarized in Table 13-7. The SPI on the saprock composite was 16.7 minutes, which would categorize this sample as very soft. The SPI's on the other samples ranged from 101.6 to 308.9 minutes, which fall in the range of hard to very hard. Orway Mineral Consultants (OMC), a subsidiary of Lycopodium, calculated the specific SAG mill grinding energy requirements determined from both the SMC and SPI test results and they found good agreement between the two methods. The results of their comparison are shown in Table 13-8.

Table 13-7: SPI Test Results

Sample ID	SPI (min)	Hardness Percentile	Category
UFZ Composite	112.2	73	Moderately Hard
Felsic Tuff	117.9	75	Hard
LFZ Composite	101.6	69	Moderately Hard
Granodiorite	115.7	75	Moderately Hard
Mafic Volcanics	269.7	96	Very Hard
Diabase	308.9	97	Very Hard
Sap Rock	16.7	5	Very Soft

Source: SGS, 2016

Table 13-8: SAG Mill Power Requirements Determined by Both SMC and SPI Test Results

Ore Types	Using SMC Results		Using SPI Results		Differences, %	
	SAG Spec. E. (kWh/t)	BM Spec. E. (kWh/t)	SAG Spec. E. (kWh/t)	BM Spec. E. (kWh/t)	SAG Spec. E.	BM Spec. E.
Felsic Tuff + FLPY	10.1	9.8	10.0	9.8	0.5	-0.5
Granodiorite + LPTF	9.9	10.6	9.8	10.7	0.8	-0.7
Mafic	14.6	12.2	14.9	11.9	-1.9	2.1
Sap Rock	3.1	6.9	3.1	5.9	1.0	13.6
Design Blend	**10.2**	**9.9**	**10.2**	**10.0**	**0.4**	**-0.4**

Source: OMC, 2016

13.3 Cyanidation Studies – Master Composites

13.3.1 Grind-Recovery Test Series

Gravity concentration tests followed by cyanidation of the gravity tailing at different grind sizes were conducted on each of the three master composites to evaluate the grind requirements. A 16 kg charge of each master composite was ground to 80% passing (P_{80}) 150 µm and then subjected to gravity concentration. The gravity separation was performed in two stages. Rougher gravity separation was conducted in a single pass using a 3 inch laboratory Knelson centrifugal concentrator operated with 1 psi fluidization water and 120 "G" force. The resulting primary gravity concentrate was subjected to one stage of upgrading on a Mozley Mineral Separator. The entire cleaned gravity concentrate was assayed for gold and silver by standard fire assay procedures to extinction.

The resulting gravity tailings were split into 2 kg test charges, then reground to target grind sizes of P_{80} 75, 90, 105, and 150 µm prior to cyanidation. Duplicate bottle roll cyanidation tests were performed at 45% solids in 0.5 grams per litre (g/L) sodium cyanide (NaCN) maintained for 72 hours at a pH of 10.5 to 11 adjusted with hydrated lime. During the leach tests, intermediate solution samples were removed to determine gold, silver, and copper dissolutions at 2, 6, 24, 30, 48, 54, and 72 hours of retention time.

The results of these tests are summarized in Table 13-9 to Table 13-11 and show that all three master composites were sensitive to grind sizes in the range of 75 to 150 µm. Gravity gold recoveries ranged from about 20% on the LFZ master composite to about 26% on the UFZ master composite. The combined gravity + cyanidation gold recoveries for UFZ, LFZ, and saprolite/saprock ranged from 91.9% to 96.2%, 92.25% to 94.9%, and 94.6% to 97.9%, respectively, at grinds ranging from P_{80} 150 to 75 µm. After review of the grind-recovery test results, a target P_{80} grind of 75 µm was selected as the optimum grind to process the Montagne d'Or ore.

Table 13-9: Summary of Gravity Concentration and Gravity Tailing Cyanidation Tests vs. Grind Size: UFZ Master Composite

Test No	Composite	Grind	NaCN	Calculated Head		Recovery						Consumption (kg/t)	
						Gravity		Cyanidation		Overall			
		P_{80} µm	(g/L)	Au (g/t)	Ag (g/t)	Au (%)	Ag (%)	Au (%)	Ag (%)	Au (%)	Ag (%)	NaCN	Ca(OH)2
GC1	UFZ Master	148	0.50	2.02	2.98	26.8	13.5	65.3	49.6	92.1	63.1	0.66	0.65
GC2	UFZ Master	148	0.50	2.05	3.08	26.3	13.1	65.4	47.9	91.7	61.0	0.70	0.61
Average	**UFZ Master**	**148**	**0.50**	**2.04**	**3.03**	**26.5**	**13.3**	**65.4**	**48.8**	**91.9**	**62.1**	**0.68**	**0.63**
GC3	UFZ Master	107	0.50	1.99	2.98	27.2	13.5	67.0	49.6	94.2	63.1	0.64	0.65
GC4	UFZ Master	107	0.50	2.05	3.08	26.3	13.1	68.1	48.0	94.4	61.1	0.67	0.53
Average	**UFZ Master**	**107**	**0.50**	**2.02**	**3.03**	**26.7**	**13.3**	**67.6**	**48.8**	**94.3**	**62.1**	**0.65**	**0.59**
GC5	UFZ Master	91	0.50	2.18	3.11	24.7	12.9	70.7	51.7	95.4	64.7	0.64	0.61
GC6	UFZ Master	91	0.50	2.13	2.98	25.4	13.5	69.5	49.6	94.8	63.1	0.64	0.61
Average	**UFZ Master**	**91**	**0.50**	**2.16**	**3.05**	**25.1**	**13.2**	**70.1**	**50.7**	**95.1**	**63.9**	**0.64**	**0.61**
GC7	UFZ Master	71	0.50	2.06	2.89	26.2	13.9	69.9	51.5	96.1	65.4	0.62	0.60
GC8	UFZ Master	71	0.50	2.15	3.11	25.1	12.9	71.2	51.7	96.3	64.7	0.63	0.58
Average	**UFZ Master**	**71**	**0.50**	**2.11**	**3.00**	**25.7**	**13.4**	**70.5**	**51.6**	**96.2**	**65.0**	**0.62**	**0.59**

Source: BV, 2016

Table 13-10: Summary of Gravity Concentration and Gravity Tailing Cyanidation vs. Grind Size: LFZ Master Composite

Test No	Composite	Grind	NaCN	Calculated Head		Recovery						Consumption (kg/t)	
						Gravity		Cyanidation		Overall			
		P_{80} µm	(g/L)	Au (g/t)	Ag (g/t)	Au (%)	Ag (%)	Au (%)	Ag (%)	Au (%)	Ag (%)	NaCN	Ca(OH)2
GC9	LFZ Master	152	0.50	1.95	6.19	20.5	5.4	71.3	42.9	91.8	48.3	0.98	0.42
GC10	LFZ Master	152	0.50	2.01	6.24	20.0	5.3	72.6	44.9	92.5	50.3	0.95	0.37
Average	**LFZ Master**	**152**	**0.50**	**1.98**	**6.21**	**20.3**	**5.4**	**71.9**	**43.9**	**92.2**	**49.3**	**0.97**	**0.40**
GC11	LFZ Master	108	0.50	1.92	6.08	20.9	5.5	71.4	41.9	92.2	47.4	0.95	0.30
GC12	LFZ Master	108	0.50	2.02	6.25	19.9	5.3	72.6	45.0	92.6	50.4	0.98	0.35
Average	**LFZ Master**	**108**	**0.50**	**1.97**	**6.16**	**20.4**	**5.4**	**72.0**	**43.5**	**92.4**	**48.9**	**0.97**	**0.33**
GC13	LFZ Master	88	0.50	2.00	6.40	20.1	5.2	73.9	46.4	94.0	51.6	0.98	0.34
GC14	LFZ Master	88	0.50	2.07	6.38	19.4	5.2	75.3	46.2	94.7	51.4	1.02	0.35
Average	**LFZ Master**	**88**	**0.50**	**2.04**	**6.39**	**19.7**	**5.2**	**74.6**	**46.3**	**94.3**	**51.5**	**1.00**	**0.34**
GC15	LFZ Master	74	0.50	1.98	6.07	20.2	5.5	75.3	48.4	95.5	53.9	0.99	0.37
GC16	LFZ Master	74	0.50	2.09	6.20	19.2	5.4	75.0	47.8	94.3	53.2	0.95	0.37
Average	**LFZ Master**	**74**	**0.50**	**2.04**	**6.13**	**19.7**	**5.4**	**75.1**	**48.1**	**94.9**	**53.5**	**0.97**	**0.37**

Source: BV, 2016

Table 13-11: Summary of Gravity Concentration and Gravity Tailing Cyanidation vs. Grind Size: Saprolite/Saprock Master Composite

Test No	Composite	Grind	NaCN	Calculated Head		Recovery						Consumption (kg/t)	
						Gravity		Cyanidation		Overall			
		P_{80} µm	(g/L)	Au (g/t)	Ag (g/t)	Au (%)	Ag (%)	Au (%)	Ag (%)	Au (%)	Ag (%)	NaCN	Ca(OH)2
GC37	Saprolite/Saprock	145	0.50	2.63	2	22.7	22.6	72.0	56.9	94.7	79.5	0.63	2.67
GC38	Saprolite/Saprock	145	0.50	2.40	2	24.9	23.9	69.7	54.3	94.6	78.3	0.69	2.67
Average		**145**	**0.50**	**2.52**	**2**	**23.8**	**23.2**	**70.8**	**55.6**	**94.6**	**78.9**	**0.66**	**2.67**
GC39	Saprolite/Saprock	102	0.50	2.49	2	24.2	22.6	71.7	57.1	96.0	79.7	0.70	2.70
GC40	Saprolite/Saprock	102	0.50	2.48	2	24.1	22.6	72.7	56.7	96.8	79.4	0.71	2.57
Average		**102**	**0.50**	**2.48**	**2**	**24.2**	**22.6**	**72.2**	**56.9**	**96.4**	**79.5**	**0.70**	**2.64**
GC41	Saprolite/Saprock	88	0.50	2.41	2	24.8	23.8	72.7	54.6	97.5	78.4	0.68	2.59
GC42	Saprolite/Saprock	88	0.50	2.97	3	20.0	21.4	78.3	59.0	98.3	80.4	0.70	2.53
Average		**88**	**0.50**	**2.69**	**2**	**22.4**	**22.6**	**75.5**	**56.8**	**97.9**	**79.4**	**0.69**	**2.56**
GC43	Saprolite/Saprock	74	0.50	2.81	3	21.2	21.4	76.6	59.2	97.9	80.5	0.74	2.52
GC44	Saprolite/Saprock	74	0.50	2.44	2	24.4	23.8	73.1	54.5	97.5	78.3	0.72	2.52
Average		**74**	**0.50**	**2.62**	**2**	**22.8**	**22.6**	**74.9**	**56.8**	**97.7**	**79.4**	**0.73**	**2.52**

Source: BV, 2016

13.3.2 Cyanide Concentration Test Series

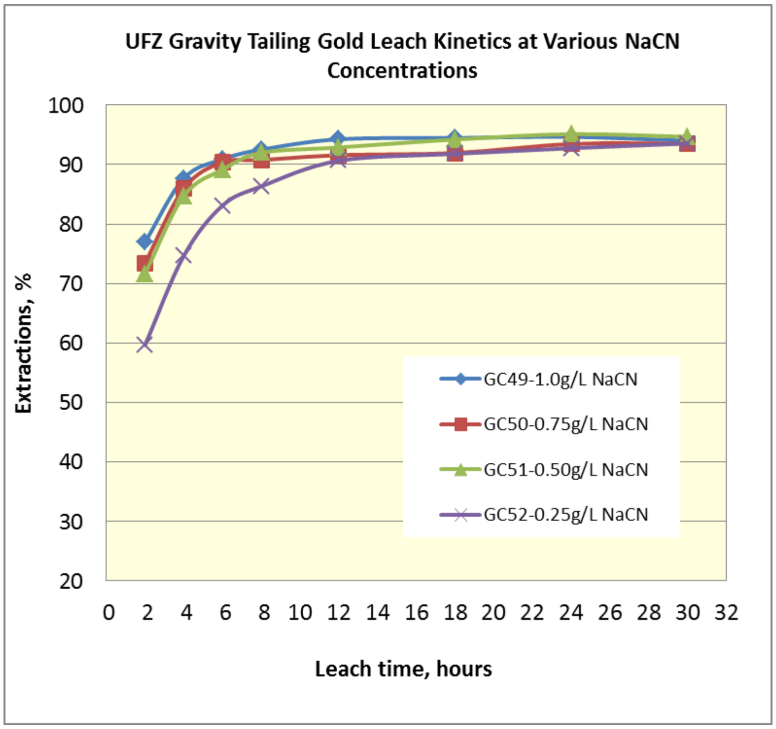
Gold extraction versus cyanide concentration tests were conducted on gravity tailings from each of the three master composites at the optimum primary grind of P_{80} 75 µm. The leach tests were conducted under optimized conditions for 30 hours at a slurry density of 45% solids with oxygen sparging to achieve 20 to 25 milligrams per litre (mg/L) $d.O_2$ in the leach slurries. Cyanide concentrations of 1.0, 0.75, 0.50 and 0.25 g/L NaCN were evaluated. The NaCN concentrations were initially set to their targets, and then allowed to attenuate to 0.2 g/L NaCN and then maintained at 0.2 g/L NaCN for the remainder of each test. Leach solution samples were removed at 2, 4, 6, 8, 12, 18, 24 and 30 hours to evaluate leach kinetics.

The results of these tests are summarized in Table 13-12 and Figure 13-3 to Figure 13-5 and demonstrated that, for the UFZ and saprolite/saprock master composites, gold extraction was not sensitive to cyanide concentration over the range tested, although increasing the initial cyanide concentration above 0.25 g/L NaCN resulted in faster leach kinetics. Overall gold recoveries of about 96% were reported for the UFZ composite and about 98% for the saprolite/saprock composite. The LFZ master composite, however, was found to be responsive to cyanide concentration, and over the concentration range tested it was found that higher cyanide dosages resulted in faster leach kinetics and higher gold extraction. Overall gold recovery from the LFZ master composite increased from 92.2% at an initial cyanide concentration of 0.25 g/L NaCN to 95% to 96% as the initial cyanide concentration was increased from 0.5 to 1.0 g/L NaCN. As a result of this test series, an initial cyanide concentration of 0.5 g/L NaCN was established as optimum.

Table 13-12: Summary of Cyanidation Tests vs. Initial Sodium Cyanide Concentration

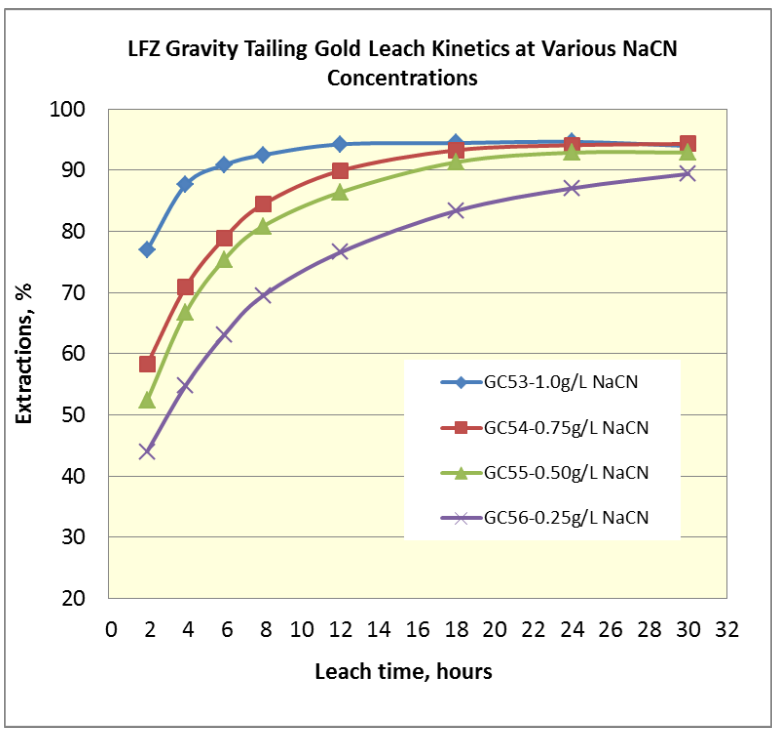
Composite	Leach Conditions			Calculated Head		Recovery						Final PLS Analysis			Residue Grade		Consumption (kg/t)	
	P_{80} Size	Average $d.O_2$	Start NaCN			Gravity		Cyanidation		Overall								
	□m	mg/L	(g/L)	Au (g/t)	Ag (g/t)	Au (%)	Ag (%)	Au (%)	Ag (%)	Au (%)	Ag (%)	Au (mg/L)	Ag (mg/L)	Cu (mg/L)	Au (g/t)	Ag (g/t)	NaCN	Ca (OH)2
UFZ Master	73	24	1.00	1.70	2.9	31.5	14.8	64.4	47.1	95.9	61.9	0.80	1.0	51	0.07	1.1	0.65	0.66
	73	23	0.75	1.70	2.9	31.6	15.0	64.0	43.1	95.6	58.1	0.79	0.9	46	0.08	1.2	0.53	0.62
	73	23	0.50	1.66	2.9	32.2	15.0	64.2	42.9	96.4	57.9	0.78	0.9	43	0.06	1.2	0.36	0.75
	73	24	0.25	1.70	3.0	31.5	14.1	64.1	40.0	95.6	54.1	0.80	0.9	44	0.08	1.4	0.28	0.75
LFZ Master	75	24	1.00	1.92	6.4	26.1	8.3	70.0	47.7	96.1	56.0	0.97	2.2	58	0.08	2.8	0.61	0.75
	75	24	0.75	1.91	6.4	26.2	8.2	69.6	45.1	95.8	53.3	0.96	2.1	56	0.08	3.0	0.53	0.77
	75	24	0.50	1.91	6.2	26.2	8.4	68.6	43.5	94.8	51.9	0.96	2.0	51	0.10	3.0	0.41	0.81
	75	23	0.25	1.87	6.1	26.7	8.7	65.5	40.2	92.2	48.9	0.90	1.8	49	0.15	3.1	0.29	0.84
Sap/Saprock Master	78	24	1.00	2.06	2.2	29.9	9.5	68.2	45.9	98.1	55.4	0.96	0.7	22	0.04	1.0	0.64	3.17
	78	24	0.75	2.05	2.4	30.0	8.9	68.5	49.0	98.5	57.9	0.96	0.8	23	0.03	1.0	0.53	3.21
	78	25	0.50	2.09	2.3	29.4	9.1	68.7	44.1	98.1	53.2	0.98	0.7	22	0.04	1.1	0.35	3.25
	78	24	0.25	2.03	2.3	30.3	9.1	67.8	43.8	98.0	52.9	0.94	0.7	21	0.04	1.1	0.26	3.26

Source: BV, 2016



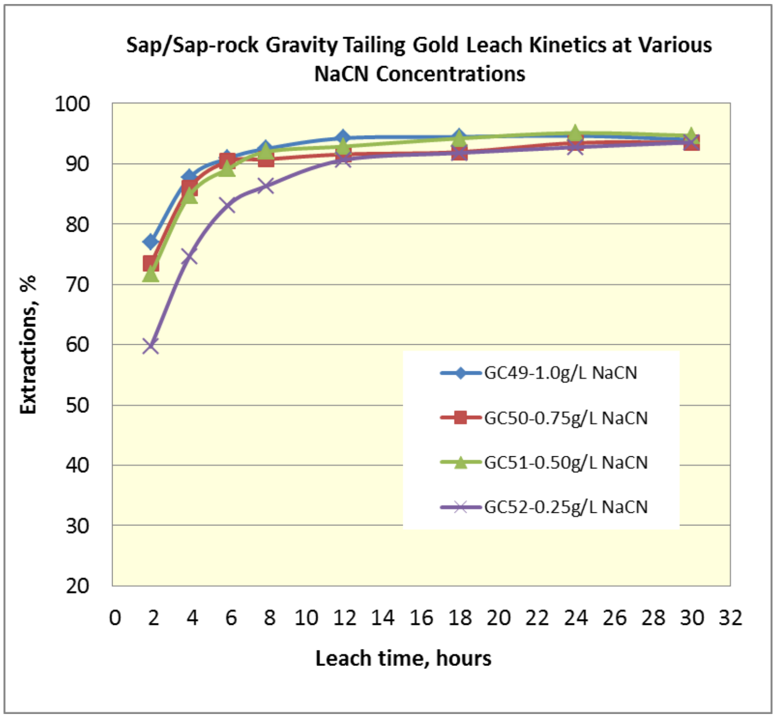
Source: BV, 2016

Figure 13-3: Gold Extraction vs. Retention Time and Cyanide Concentration: UFZ Gravity Tailing



Source: BV, 2016

Figure 13-4: Gold Extraction vs. Retention Time and Cyanide Concentration: LFZ Gravity Tailing



Source: BV, 2016

Figure 13-5: Gold Extraction vs. Retention Time and Cyanide Concentration: Saprolite/Saprock

13.3.3 Large Batch Confirmation Tests

Large batch tests were conducted on 20 kg test charges of the UFZ, LFZ, saprolite/saprock master composite and the master composite Blend composite (1/3 UFZ + 1/3 LFZ + 1/3 saprolite/saprock) to confirm the gravity + cyanidation process flowsheet using the optimized test conditions established for the master composites. These tests were conducted at a target P_{80} grind of 75 µm. Gravity concentration was performed in two stages, and cyanidation of the gravity tailings was carried out in duplicate for 30 hours at 50% solids (45% solids for saprolite master composite). Cyanide concentration was initially adjusted to 0.5 g/L NaCN and allowed to attenuate to 0.2 g/L NaCN. Oxygen sparging was used to maintain 20 to 25 mg/L $d.O_2$ in the leach slurry. The results of these tests are summarized in Table 13-13 and confirm the amenability of the Montagne d'Or master composites to the proposed gravity + cyanidation process at a larger scale. The overall gold recoveries ranged from 96.4% to 97.5% and overall silver recoveries ranged from 56.6% to 73.3%. Cyanide consumption averaged 0.42 kilograms per tonne (kg/t) NaCN for the UFZ, LFZ and master composite blend and 0.55 kg/t NaCN for the saprolite/saprock master composite. Lime consumption was almost 0.5 kg/t for the UFZ and LFZ master composites and was significantly higher at about 2.5 kg/t for the saprolite/saprock master composite. Copper in the pregnant leach solution was relatively low and ranged from 20 to 58 mg/L. Gold leaching is essentially complete after 24 hours of retention time.

Table 13-13: Summary of Large Batch Test Work on Montagne d'Or Master Composites

Composite	Test No	Leach Conditions				Calculated Head		Recovery (%)						Consumption (kg/t)	
		P_{80} Size	Slurry Density	Average $d.O_2$	Start NaCN			Gravity		Cyanidation		Overall			
		µm	%	mg/L	(g/L)	Au (g/t)	Ag (g/t)	Au	Ag	Au	Ag	Au	Ag	NaCN	Ca(OH)2
UFZ Master	BGC1	74	50	23	0.50	1.90	2.6	37.1	23.1	59.7	50.2	96.8	73.3	0.43	0.44
LFZ Master	BGC2	75	50	23	0.50	2.07	5.3	31.3	11.5	65.0	45.5	96.4	57.0	0.43	0.46
Saprolite Master	BGC3	77	45	22	0.50	2.02	2.3	35.4	17.1	62.1	39.5	97.5	56.6	0.55	2.56
Composite Blend	BGC4	74	50	21	0.50	1.94	3.8	30.3	12.0	67.1	40.2	97.4	52.2	0.42	0.32

Source: BV, 2016

13.4 Cyanidation Studies – Variability Composites

Confirmatory testing of the process flowsheet using the optimized test conditions established for the master composites, was conducted on the seven variability composites representing seven mine phases and on 15 variability composites representing the major lithologies and grade ranges. These tests were conducted at a target grind size of P_{80} 75 µm. Gravity concentration was performed in two stages, and cyanidation of the gravity tailing was carried out in duplicate for 30 hours at 50% solids (45% solids for saprolite samples). NaCN concentration was initially adjusted to 0.5 g/L NaCN and allowed to attenuate to 0.2 g/L NaCN. Oxygen sparging was used to maintain 20 to 25 mg/L d.O_2 in the leach slurry.

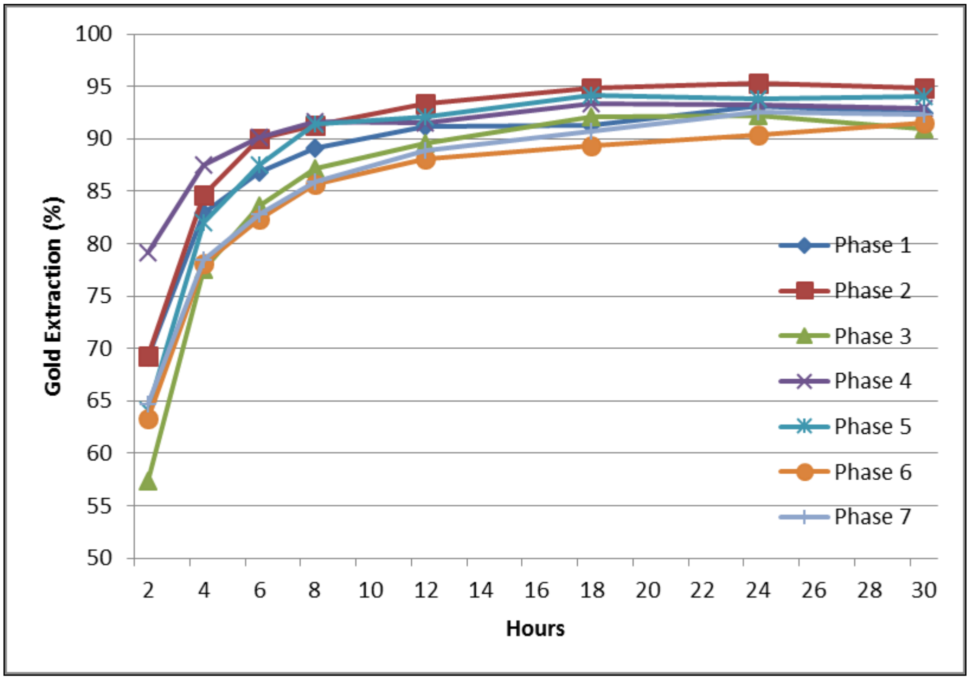
13.4.1 Mine Phase Variability Composites

The results of tests conducted on the mining phase variability composites are summarized in Table 13-14. Overall gold recovery was fairly consistent, ranging from 93.9% for the Phase 6 composite to 96.6% for the Phase 5 composite. Silver recovery was more variable and ranged from 36.1% for the Phase 7 composite to 73.2% for the Phase 5 composite. Cyanide consumption was similar to the master composites and ranged from 0.34 to 0.48 kg/t. Gold leach kinetics are shown in Figure 13-6 and confirm that leaching is essentially complete after about 24 hours.

Table 13-14: Summary of Gravity + Cyanidation Tests on the Mining Phase Variability Composites

Composite	Leach Conditions				Calculated Head		Recovery (%)						Residue Grade		Consumption (kg/t)	
	P_{80} Size	Average $d.O_2$	Slurry Density	Start NaCN			Gravity		Cyanidation		Overall					
	□m	mg/L	%	(g/L)	Au (g/t)	Ag (g/t)	Au	Ag	Au	Ag	Au	Ag	Au (g/t)	Ag (g/t)	NaCN	Ca(OH)2
Mining Phase 1	77	23	50	0.50	2.68	2.7	43.8	6.1	51.7	47.9	95.4	53.9	0.12	1.25	0.46	0.63
Mining Phase 2	80	22	50	0.50	2.38	3.3	26.2	28.6	70.1	44.1	96.3	72.7	0.09	0.90	0.48	1.10
Mining Phase 3	78	23	50	0.50	2.11	8.6	47.5	6.8	47.7	41.8	95.3	48.6	0.10	4.45	0.42	0.55
Mining Phase 4	79	22	50	0.50	2.34	1.7	42.3	0.8	53.6	58.9	95.9	59.6	0.10	0.70	0.39	0.44
Mining Phase 5	78	22	50	0.50	2.78	3.7	39.5	19.9	57.1	53.3	96.6	73.2	0.10	1.00	0.34	0.71
Mining Phase 6	75	22	50	0.50	1.56	2.8	32.1	5.1	61.8	42.5	93.9	47.6	0.10	1.45	0.37	0.63
Mining Phase 7	75	22	50	0.50	2.03	3.4	32.7	6.9	62.5	29.2	95.2	36.1	0.10	2.20	0.34	0.45

Average of duplicate tests on each composite
Source: BV, 2016



Source: BV, 2016

Figure 13-6: Gold Leach Kinetics for Mining Phase Variability Composites

13.4.2 Ore Lithology Variability Composites

The results of tests conducted on the ore lithology variability composites for several different grade ranges are summarized in Table 13-15. Average overall gold recovery was fairly consistent for the saprolite, saprock, granodiorite and felsic tuff lithologies and ranged from about 96% to 98%. Average gold recovery from the mafic volcanic lithology was lower at about 93%. Gold recovery appeared to be insensitive to ore grade over the range tested.

Average silver recovery was more variable and ranged from 40% for the mafic volcanic lithology to 72% for the saprock lithology. Silver recovery versus silver grade within each lithology was also variable, and did not necessarily correlate with grade. Average cyanide consumption consistent for the lithologies tested and ranged from 0.33 to 0.39 kg/t. Gold and silver leach kinetics were rapid and similar to the leach kinetics observed for the mining phase variability composites.

Table 13-15: Summary of Gravity + Cyanidation Tests on the Lithology Variability Composites [(1)]

Composite	Leach Conditions				Calc. Head [(2)]		Recovery (%)						Residue Grade		Consumption (kg/t)	
	P_{80} Size	Average $d.O_2$	Slurry Density	Start NaCN			Gravity		Cyanidation		Overall					
	µm	mg/L	%	(g/L)	Au (g/t)	Ag (g/t)	Au	Ag	Au	Ag	Au	Ag	Au (g/t)	Ag (g/t)	NaCN	Ca(OH)2
Granodiorite – Average Grade	74	22	50	0.50	2.90	4.6	49.9	18.1	46.8	38.5	96.7	56.5	0.10	2.00	0.31	0.65
Granodiorite – Higher Grade	80	23	50	0.50	2.35	4.7	41.6	12.8	54.9	52.2	96.5	65.0	0.08	1.65	0.35	0.51
Granodiorite – Lower Grade	72	23	50	0.50	2.55	1.9	64.7	19.5	33.7	41.1	98.3	60.7	0.04	0.75	0.34	0.63
Saprolite – Average Grade	79	24	44	0.50	2.51	2.3	50.5	16.9	47.8	44.0	98.3	60.9	0.04	0.90	0.39	2.27
Saprolite – Higher Grade	71	22	45	0.50	1.17	0.8	24.2	2.2	74.8	34.8	98.9	37.0	0.01	0.50	0.35	2.15
Saprolite – Lower Grade	71	23	44	0.50	1.48	1.7	47.5	15.4	50.5	34.5	98.0	50.0	0.03	0.85	0.30	1.89
Saprock – Average Grade	74	23	45	0.50	2.07	1.6	51.7	27.6	46.3	57.2	98.1	84.8	0.04	0.25	0.41	1.82
Saprock – Higher Grade	77	24	44	0.50	3.31	2.3	40.1	22.4	57.5	40.0	97.7	62.4	0.08	0.85	0.41	1.82
Saprock – Lower Grade	77	23	44	0.50	1.85	2.6	47.7	11.7	50.7	57.1	98.4	68.8	0.03	0.80	0.36	1.59
Felsic Tuff – Average Grade	78	23	50	0.50	2.02	3.5	37.8	9.9	57.9	42.9	95.7	52.7	0.09	1.65	0.39	0.46
Felsic Tuff – Higher Grade	74	21	51	0.50	2.63	4.4	44.7	12.5	51.1	48.6	95.8	61.1	0.11	1.70	0.32	0.67
Felsic Tuff – Lower Grade	74	22	50	0.50	1.18	2.2	37.5	10.2	58.3	48.7	95.8	58.9	0.05	0.90	0.29	0.60
Mafic Volcanics – Average Grade	79	21	51	0.50	2.21	4.1	32.6	4.2	60.0	26.6	92.5	30.8	0.17	2.85	0.40	0.77
Mafic Volcanics – Higher Grade	79	23	50	0.50	2.75	2.8	26.2	7.7	65.2	30.7	91.5	38.4	0.24	1.75	0.41	0.71
Mafic Volcanics – Lower Grade	79	22	50	0.50	1.65	5.7	28.7	7.0	66.7	43.9	95.5	50.9	0.08	2.80	0.37	0.70

(1) Average of duplicate tests on each composite
(2) Gold grades don't consistently correspond to target grade range due to presence of coarse gold
Source: BV, 2016

13.5 Gravity Recoverable Gold Test Work and Modeling

Extended Gravity Recoverable Gold (EGRG) tests were conducted by FLSmidth (Knelson) on the UFZ, LFZ and saprolite/saprock master composites and the results of these tests were used to model the gravity circuit for the Montagne d'Or process plant. The Gravity Recoverable Gold (GRG) value of an ore sample provides an indication of the amenability of the ore sample to gravity concentration. The GRG testing procedure is based on progressive particle size reduction, which allows recovery of gold as it becomes liberated while minimizing over grinding. The GRG test consists of sequential liberation and recovery stages. The results of the EGRG tests on each composite are presented in Table 13-16 and show EGRG values that range from 52.6 for the saprolite/saprock master composite to 63.5 for the UFZ composite at the target grind size of P_{80} 75 µm (actual grinds ranged from P_{80} 73 to 91 µm). The EGRG values were used by FLSmidth to model the gravity circuit for the Montagne d'Or process plant.

Table 13-16: Summary of EGRG Tests on Montagne d'Or Master Composites

Composite	Head Grade (Au g/t)	Final Tails (Au g/t)	EGRG Value	Stage Recoveries			Final Grind Size (P_{80} µm)	AMIRA Classification
				1st	2nd	3rd		
LFZ	1.8	0.8	55.6	18.0	28.1	9.5	91	Moderate to Coarse
SAP	2.1	1.0	52.6	25.0	11.2	16.3	73	Coarse
UFZ	1.7	0.6	63.5	21.2	31.0	11.2	82	Moderate to Coarse

Source: FLSmidth, 2016

13.5.1 Gravity Circuit Modeling

Gravity circuit modeling was conducted by FLSmidth using Knelson's size-by-size mathematical model to predict gold recovery within the grinding circuit. The model input data provided by Lycopodium included:

- Ore feed rate: 563 t/h;
- Circulating load: 200% to 400%; and
- Grind size: P_{80} 75 µm.

The results of this simulation are presented in Table 13-17, which show predicted gold recoveries of about 38% for the UFZ and LFZ master composites and about 31% for the saprolite/saprock master composite when the grinding circuit is operated with a 250% circulating load. This represents a recovery of about 60% of the gravity recoverable gold. Predicted gold recoveries are incrementally lower when the grinding circuit is operated at a 400% circulating load. Based on these results, FLSmidth recommended two KC-QS48 Knelson centrifugal concentrators for the application and concluded that if the concentrators were operated with G6 cones and with 45 to 60 minute concentrating cycle times, concentrate production will be in the range of 3,000 to 4,400 kg/day. This would require a Consep CS3000 Acacia Intensive Leach Reactor to leach the gravity concentrate.

Table 13-17: Gravity Circuit Modeling Results

Sample	Circulating Load (%)	Au Recovery	
		(%)	(% GRG)
LFZ	250	37.2	61.6
SAP	250	31.5	59.8
UFZ	250	38.9	60.3
LFZ	400	32.8	54.3
SAP	400	27.9	53.0
UFZ	400	34.4	53.4

Source: FLSmidth, 2016

13.5.2 Gravity Concentrate Leaching

Intensive cyanide leach (ICL) tests were conducted by FLSmidth on each of the gravity concentrates produced from the EGRG test work in order to establish the extent to which the gold contained in the gravity concentrate could be recovered by cyanidation. Each ICL test was conducted on about 300 g of gravity concentrate in a 20 g/L sodium cyanide solution at 40% solids for 24 hours. LeachAid™ was added at a rate of 7 g per 100 g of concentrate, dry weight. Lime was added to adjust the pH above 10.5. The results of these tests are summarized in Table 13-18 and demonstrate that 98% to 99% of the gold contained in the gravity concentrates could be extracted. Sodium cyanide consumption ranged from 36.1 to 39.7 kg/t of concentrate (equivalent to about 0.4 to 0.5 kg/t ore). The gravity concentrates represented about 1.0 to 1.5 wt% of the ore. Generally, gold extraction was complete within 8 hours of intensive leaching. These tests demonstrated that gold contained in the gravity concentrate could be nearly completely extracted by cyanidation, but leach conditions were not optimized and reported cyanide consumptions were substantially higher than will likely be experienced in the process plant.

Table 13-18: Summary of Intensive Leach Tests

LFZ Gravity Concentrate		
Time (hrs)	**Au Assay (mg/L)**	**Au Extraction (%)**
0	0	0
2	45.6	100
4	40.5	92.9
8	38.7	92.1
24	40.9	98.9
Residue	0.71	1.1
Calc. Head (g/t)		80.8
NaCN Consumption (kg/t conc)		36.1
Saprolite/Saprock Gravity Concentrate		
Time (hrs)	**Au Assay (mg/L)**	**Au Extraction (%)**
0	0	0
2	56.2	98.5
4	53.1	97.2
8	54.6	100
24	52.8	99.5
Residue	0.44	0.5
Calc. Head (g/t)		73.3
NaCN Consumption (kg/t conc)		39.7
UFZ Gravity Concentrate		
Time (hrs)	**Au Assay (mg/L)**	**Au Extraction (%)**
0	0	0
2	40.4	91.6
4	43.9	100
8	42.6	100
24	39.5	98.4
Residue	1.03	1.6
Calc. Head (g/t)		77.2
NaCN Consumption (kg/t conc)		36.7

Source: FLSmidth, 2016

13.6 Detoxification Studies

Cyanide detoxification test work using the industry-standard SO_2/Air process was conducted by SGS Lakefield (SGS) as part of a confirmatory test program that included bulk cyanidation tests on gravity tailings produced from the UFZ and saprolite/saprock master composites using optimized test conditions established from the test program conducted by BV. The test program at SGS also included carbon adsorption test work and modeling followed by cyanide detoxification studies on the barren leach solution. SGS conducted batch detoxification tests on each cyanidation tailing sample to first establish the approximate detoxification conditions and to generate a treated product with low residual cyanide for used as starting material for the continuous detoxification tests on each composite. Three continuous detoxification tests were run on cyanidation tailings produced from both the UFZ and saprolite/saprock master composites, and the results are summarized in Table 13-19. These tests demonstrated that cyanide in the leach residue could readily be detoxified to less than 1 ppm CN_{wad}. SO_2 consumption in the range of about 5 – 6 g SO_2/g CN_{wad} were reported, which is typical of industry practice.

Table 13-19: Summary of Cyanide Detoxification Tests on Cyanidation Residues from UFZ and Saprolite/Saprock Master Composites

Test	Composite	Time (min.)	pH	Detox Feed and Effluent					Reagent Addition			Reagent Addition (kg/t solids)		
				CN_T	CN_{WAD}		Cu	Fe	(g/g CNWAD)			(kg/t solids)		
				(mg/L)	Lab (mg/L)	PA (mg/L) 1	(mg/L)	(mg/L)	SO2 (Equiv)	Lime	Cu	SO2 (Equiv)	Lime	Cu
Detox Feed			9.7	150	129		114	2.96						
CND 1-1	UFZ	59	8.6			15.2			6.40	4.66	0	0.82	0.60	0
CND 1-2	UFZ	55	8.6			7.26			7.87	6.14	0	1.01	0.79	0
CND 1-3	UFZ	54	8.5	<0.1	<0.1	<0.1	0.8	<0.1	5.95	4.69	0.16	0.77	0.60	0.02
Detox Feed			10.3	151	133		38.9	<0.05						
CND 2-1	Sap/Saprock	57	9.1			1.18			5.24	0.02	0	0.85	0	0
CND 2-2	Sap/Saprock	60	9.0			1.41			5.73	0	0	0.93	0	0
CND 2-3	Sap/Saprock	58	9.0	0.60	<0.1	1.26	0.3	0.2	5.40	0	0.08	0.88	0	0.01
CND 2-4	Sap/Saprock	58	8.8	0.58	0.23	1.05	0.4	0.2	5.46	0	0.15	0.89	0	0.02

Note: PA = Picric acid method
Source: SGS, 2016

13.7 Thickening and Rheological Studies

Thickening and rheological studies were conducted on selected master composites which had been ground to the target grind of P_{80} 75 µm in order to obtain the necessary design parameters to size the grinding control and tailing wash thickeners that will be included in the Montagne d'Or process flowsheet. Test work was conducted both by Pocock Industrial (Pocock), a respected company that specializes in solid liquid separation technologies, and by Outotek. The results of this work are presented in this section.

13.7.1 Pocock Industrial Thickening and Rheological Studies

Pocock conducted static and continuous thickening tests on the UFZ and saprolite/saprock master composite and the master composite Blend (1/3 UFZ + 1/3 LFZ + 1/3 sap/saprock) to determine conventional and high rate thickener requirements. Each composite was ground to the target primary grind of P_{80} 75 µm by BV Minerals in Richmond, British Columbia and shipped to Pocock Industrial's laboratory in Salt Lake City, Utah. Upon receipt, the pH of each composite material was adjusted to pH 10.5 with lime and maintained at this target pH throughout the entire test program.

Dynamic thickening tests were performed on each composite to determine the recommended maximum hydraulic design basis for high rate thickener design. Expected underflow slurry density and overflow suspended solids concentrations were also determined, and the results of the dynamic test work are presented in Table 13-20. The recommended design hydraulic loading rates generally ranged from 3.2 to 4.5 m^3/m^2 h at the feed density and flocculant dosages tested. Predicted maximum underflow slurry densities generally ranged from 63% to 70% solids for the Blend and UFZ composites. The predicted underflow density for the saprolite/saprock composite was significantly lower at 55% solids. Overflow clarities were generally acceptable and ranged from 150 to 250 mg/L suspended solids. All three composites required flocculant dosages slightly higher than required for the conventional thickener based on static test results (approximately 5 to 10 g/t additional flocculant for each material).

Table 13-20: Recommended High Rate Thickener Operating Parameters

Material Tested	Recommended High Rate Thickener Operating Parameter Ranges						
	Tested Feed Solids (%)	Flocculant Type	Flocculant Dose (g/t)	Flocculant Conc. (g/l)	Design Basis Net Feed Loading (m^3/m^2 hr)	Predicted Overflow TSS Conc. Range (mg/l)	Predicted Underflow Density
UFZ Master Composite	20.4	Hychem AF304	32 – 38	0.1 – 0.2	4.45	150 – 250	70.0%
UFZ / LFZ / Saprolite Sap-Rock Blend Composite	13.6	Hychem AF304	19 – 27	0.1 – 0.2	3.26	150 – 250	63.0%
Saprolite / Sap-Rock Master Composite	14.9	Hychem AF304	19 – 24	0.1 – 0.2	3.25	150 – 250	55.0%

Source: Pocock 2016

13.7.2 Outotek Thickening Studies

Outotek conducted confirmatory dynamic thickening tests on the gravity tailings generated at the primary target grind of P_{80} 75 µm by BV from the UFZ master composite and the Blend master composite sent to Outotek's Sudbury laboratory for testing. The test program was conducted according to the following parameters:

- Solids feed rate: 563 t/h;
- Feed slurry density: 30% w/w;
- Slurry pH: 10.5;
- Particle size: P_{80} 74 µm;
- Underflow density: 55% to 65%; and
- Overflow clarity: 150 to 200 mg/L TSS.

Flocculant screening tests were conducted on both composites and it was determined that Flomin's SNF 910 flocculant offered the best overall performance at a dosage of 15 g/t. The results of dynamic thickening tests conducted on the UFZ master composite are shown in Table 13-21 and the test results on the Blend master composite are shown in Table 13-22.

Table 13-21: Summary of Outotec Dynamic Thickening Tests on the UFZ Master Composite

Run No.	Feed		Flocculant		Underflow		Overflow	Solids SG
	SLR	Rise Rate	Type	Dose	Meas. Solids	YS	Solids	(Measured)
	($t/(m^2 h)$)	(m/h)		(g/t)	(% (w/w))	(Pa)	(ppm)	(t/m^3)
1	1.50	5.16	Flomin 910 VHM	10	60.9	30	28	2.81
2	1.50	5.16	Flomin 910 VHM	5	60.6	26	73	2.83
3	2.00	6.88	Flomin 910 VHM	5	59.7	13	95	2.83
4	2.50	8.60	Flomin 910 VHM	5	56.9	9	179	2.83
5	1.50	5.16	Flomin 910 VHM	3	60.0	8	359	2.84
6	2.50	8.60	Flomin 910 VHM	3	55.6	6	247	2.88
7	1.50	4.59	Flomin 910 VHM	5	60.8	12	198	2.83

Source: Outotec, 2016

Table 13-22: Summary of Outotec Dynamic Thickening Tests on the Blend Master Composite

Run No.	Feed		Flocculant		Underflow		Overflow	Solids SG
	SLR	Rise Rate	Type	Dose	Meas. Solids	YS	Solids	(Measured)
	($t/(m^2 h)$)	(m/h)		(g/t)	(% (w/w))	(Pa)	(ppm)	(t/m^3)
1	1.50	5.28	SNF 910	5	56.6	16	35	2.78
2	1.50	5.28	SNF 910	10	59.5	28	22	2.91
3	1.50	5.28	SNF 910	15	59.8	34	18	2.90
4	2.00	7.04	SNF 910	10	58.4	28	31	2.95
5	2.00	7.04	SNF 910	5	56.7	16	42	2.91
6	2.50	8.79	SNF 910	10	54.9	21	36	2.87
7	2.00	6.01	SNF 910	5	55.6	19	24	2.80

Source: Outotec, 2016

13.8 Recoverability

The results of gravity + cyanidation tests conducted on each of the master composites and a master composite blend under optimized test conditions are summarized in Table 13-23. Overall gold recovery ranged from 94.8% to 98.1%. In order to allow for inherent plant inefficiencies, SRK has adjusted these reported recoveries down by 2% resulting in adjusted gold recoveries ranging from 93% to 96% and averaging 94%. Adjusted silver recovery ranged from 47% to 54% and averaged 50%.

The results of gravity + cyanidation tests conducted on each of the lithology variability composites under optimized test conditions are summarized in Table 13-24. Overall gold recovery ranged from 93.1% to 98.4%. In order to allow for inherent plant inefficiencies, SRK has adjusted these reported recoveries down by 2% resulting in adjusted gold recoveries ranging from 91% to 96% and averaging 94%. Adjusted silver recovery ranged from 38% to 70% and averaged 54%.

SRK has estimated overall adjusted gold and silver recoveries based on the contribution from each ore lithology during each phase of mining. These recovery estimates are presented in Table 13-25. During the first six mining phases gold recovery is estimated at 94% to 95% and silver recovery is estimated at about 54% to 56%. Gold is projected to decline slightly during the final phase of mining when ore will be derived primarily from a combination of felsic tuff and mafic lithologies. Estimated reagent consumptions are also shown in Table 13-25.

Table 13-23: Summary of Gravity Concentration and Cyanidation on Montagne d'Or Master Composites

Master Composites	Test	Calc. Head		Gravity Recovery (%)		Cyanidation Extr. (%)		Overall (%)		Adjustment	Adjusted Recovery %		Consumption (kg/t)	
		Au (g/t)	Ag (g/t)	Au	Ag	Au	Ag	Au	Ag	Factor [1]	Au	Ag	NaCN	Ca(OH)2
UFZ	GC-63	1.73	3.0	30.9	14.5	65.1	41.5	96.0	56.0	2.0	94.0	54.0	0.30	0.70
LFZ	GC-67	1.90	6.1	26.2	8.6	68.6	40.6	94.8	49.2	2.0	92.8	47.2	0.30	0.79
Saprolite/Saprock	GC-71	2.06	2.3	29.1	9.0	69.0	43.3	98.1	52.3	2.0	96.1	50.3	0.30	3.00
Blend [2]		1.90	3.8	28.7	10.7	67.6	41.8	96.3	52.5	2.0	94.3	50.5	0.30	1.50

Source: BV and SRK, 2016
Notes:
(1) Laboratory recoveries adjusted down by 2% to account for inherent plant inefficiencies
(2) Calculated blend based on 1/3 UFZ + 1/3 LFZ + 1/3 Saprolite/Saprock

Table 13-24: Summary of Gravity Concentration and Cyanidation on Lithology Composites

Lithology Composites	Test	Calc. Head		Gravity Recovery (%)		Cyanidation Extr. (%)		Overall (%)		Adjustment	Adjusted Recovery (%)		Consumption (kg/t)	
		Au (g/t)	Ag (g/t)	Au	Ag	Au	Ag	Au	Ag	Factor (1)	Au	Ag	NaCN	Ca(OH)2
Saprolite														
Low Grade	VT-12	1.17	0.8	24.2	2.2	74.8	34.8	99.0	37.0	2.0	97.0	35.0	0.35	2.15
Medium Grade	VT-13	1.48	1.7	47.5	15.4	50.5	34.5	98.0	49.9	2.0	96.0	47.9	0.30	1.89
High Grade	VT-11	2.51	2.3	50.5	16.9	47.8	44.0	98.3	60.9	2.0	96.3	58.9	0.39	2.27
Average		**1.72**	**1.6**	**40.7**	**11.5**	**57.7**	**37.8**	**98.4**	**49.3**	**2.0**	**96.4**	**47.3**	**0.35**	**2.10**
Sap-rock														
Low Grade	VT-16	1.85	2.6	47.7	11.7	50.7	57.1	98.4	68.8	2.0	96.4	66.8	0.36	1.59
Medium Grade	VT-14	2.07	1.6	51.7	27.6	46.3	57.2	98.0	84.8	2.0	96.0	82.8	0.41	1.82
High Grade	VT-15	3.31	2.3	40.1	22.4	57.5	40.0	97.6	62.4	2.0	95.6	60.4	0.41	1.82
Average		**2.41**	**2.2**	**46.5**	**20.6**	**51.5**	**51.4**	**98.0**	**72.0**	2.0	**96.0**	**70.0**	**0.39**	**1.74**
Felsic Tuff														
Low Grade	VT-19	1.18	2.2	37.5	10.2	58.3	48.7	95.8	58.9	2.0	93.8	56.9	0.29	0.60
Medium Grade	VT-17	2.02	3.5	37.8	9.9	57.9	42.9	95.7	52.8	2.0	93.7	50.8	0.39	0.46
High Grade	VT-18	2.63	4.4	44.7	12.5	51.1	48.6	95.8	61.1	2.0	93.8	59.1	0.32	0.67
Average		**1.94**	**3.4**	**40.0**	**10.9**	**55.8**	**46.7**	**95.8**	**57.6**	2.0	**93.8**	**55.6**	**0.33**	**0.58**
Granodiorite														
Low Grade	VT-9	2.35	4.7	41.6	12.8	54.9	52.2	96.5	65.0	2.0	94.5	63.0	0.35	0.51
Medium Grade	VT-10	2.55	1.9	64.7	19.5	33.7	41.1	98.4	60.6	2.0	96.4	58.6	0.34	0.63
High Grade	VT-8	2.90	4.6	49.9	18.1	46.8	38.5	96.7	56.6	2.0	94.7	54.6	0.31	0.65
Average		**2.60**	**3.7**	**52.1**	**16.8**	**45.1**	**43.9**	**97.2**	**60.7**	**2.0**	**95.2**	**58.7**	**0.33**	**0.60**
Lithology Composites	**Test**	**Calc. Head**		**Gravity Recovery (%)**		**Cyanidation Extr. (%)**		**Overall (%)**		**Adjustment**	**Adjusted Recovery (%)**		**Consumption (kg/t)**	
		Au (g/t)	**Ag (g/t)**	**Au**	**Ag**	**Au**	**Ag**	**Au**			**Au (g/t)**	**Ag (g/t)**	**Au**	**Ag**
Mafic														
Low Grade	VT-22	1.65	5.7	28.7	7.0	66.7	43.9	95.4	50.9	2.0	93.4	48.9	0.37	0.70
Medium Grade	VT-20	2.21	4.1	32.6	4.2	60.0	26.6	92.6	30.8	2.0	90.6	28.8	0.40	0.77
High Grade	VT-21	2.75	2.8	26.2	7.7	65.2	30.7	91.4	38.4	2.0	89.4	36.4	0.41	0.71
Average		**2.20**	**4.2**	**29.2**	**6.3**	**64.0**	**33.7**	**93.1**	**40.0**	2.0	**91.1**	**38.0**	**0.39**	**0.73**

recoveries adjusted down by 2% to account for inherent plant inefficiencies
Source: BV and SRK 2016

Table 13-25: Estimated Recoveries by Mining Phase Based on Lithology Contribution

Lithology	Phase 1	Phase 2	Phase 3	Phase 4	Phase 5	Phase 6	Phase 7
	Wt (%)	Wt (%)	Wt (%)	Wt (%)	Wt (%)	Wt (%)	Wt (%)
Saprolite	18	34	15	6	8	3	3
Sap-Rock	5	11	7	2	2	1	1
Felsic Tuff	70	53	69	80	76	85	52
Granodiorite	7	2	0	12	14	3	0
Mafic	0	0	9	0	0	8	44
Au Recovery (%)	94.5	94.9	94.1	94.1	94.2	93.7	92.7
Ag Recovery (%)	55.0	54.4	53.8	55.8	55.7	54.2	47.8
Reagent Consumption							
NaCN (kg/t)	0.34	0.34	0.34	0.34	0.34	0.34	0.36
Ca(OH)2 (kg/t)	0.91	1.22	0.90	0.69	0.72	0.65	0.70
Flocculant (1)							
Pre-leach Thickener (g/t)	25	25	25	25	25	25	25
Tails Wash Thickener (g/t)	25	25	25	25	25	25	25

Hychem 308, or equivalent, for High Rate thickener application
Source: BV and SRK 2016

13.9 Significant Factors

The following significant factors are identified based on the metallurgical studies conducted for the BFS:

- The BFS metallurgical program focused on the development of a process flowsheet that included gravity concentration followed by cyanidation of the gravity tailings and intensive cyanide leaching of the gravity concentrate.
- The metallurgical program was conducted on three master composites, 15 variability composites representing different ore lithologies and grade ranges, and 7 variability composites representing seven mining phases that were identified at the start of the program.
- Montagne d'Or ore can be readily processed to recover the contained gold and silver values using unit operations considered standard to the industry.
- SRK has estimated overall adjusted gold and silver recoveries based on the contribution from each ore lithology during each phase of mining. During the first six mining phases gold recovery is estimated at 94% to 95% and silver recovery is estimated at about 54% to 56%. These recovery projections include a 2% deduction from reported laboratory test results to account for inherent plant inefficiencies.
- Detoxification of the cyanide leach residues was accomplished with the industry-standard SO_2/Air process. It was demonstrated that cyanide in the leach residue could readily be detoxified to less than 1 ppm CN_{wad}. SO_2 consumption in the range of about 5 – 6 g SO_2/g CN_{wad} were reported, which is typical of industry practice.

14 Mineral Resource Estimate

14.1 Basis of Resource Estimation

The mineralization at Montagne d'Or is valued primarily for its gold content. There are however, localized zones with copper mineralization. Only gold grades were estimated in the work described in this report because the low levels of copper do not support economic extraction.

Dr. Bart Stryhas constructed the geologic and Mineral Resource model discussed below. He is responsible for the resource estimation methodology, Mineral Resource classification and resource statement. Dr. Stryhas is independent of the issuer applying all of the tests in Section 1.5 of NI 43-101.

The resource estimation is based on the current drillhole database, interpreted lithologies, geologic controls and current topographic data. The resource estimation is supported by drilling and sampling current to July 1, 2016. The estimation of Mineral Resources was completed utilizing computerized resource block model constructed using Vulcan™ modeling software.

14.2 General Geology and Geologic Model

The Montagne d'Or deposit is a Paleoproterozoic age gold deposit that has undergone remobilization and shear zone style deformation. The deposit is located within the northern greenstone belt of the Guiana Shield. Mineralization is hosted within the two billion year old, Paramaca Formation composed predominantly of metavolcanic and metasedimentary units. These units have been deformed by high strain isoclinal folding and ductile shearing which has developed a pervasive foliation striking east-west and dipping steeply to the south. The current model of gold mineralization is a VMS type. Significant portions of the deposit are thought to have been emplaced as replacement style mineralization. Subsequently, the mineralization has been deformed and partly remobilized within structural controls. Gold mineralization is associated with primary sulphide minerals as replacements within pyrite and chalcopyrite. At a macroscopic scale, the following five types of mineralization have been identified in mapping and drill core logging:

- SMS (>20% sulphides) with associated gold mineralization;
- Sulphides as disseminations and stringers with associated gold mineralization;
- Late-stage disseminated euhedral pyrite mineralization;
- Rhythmic mafic tuff with associated pyrrhotite mineralization; and
- Gold mineralization associated with quartz veins.

The mineralization is hosted within a tightly to isoclinally folded, steeply south dipping lithological package consisting of felsic and mafic metavolcanic rocks cut by felsic and mafic intrusives. The felsic metavolcanic rocks are the largest mineralized unit followed by the mafic metavolcanic rocks. The felsic metavolcanic rocks are subdivided into a felsic tuff and lapilli tuff. The metavolcanic rocks are intruded by three distinct felsic to intermediate plutonic units that host minor amounts of mineralization; from oldest to youngest these are granodiorite, quartz-feldspar porphyry and feldspar porphyry. All units described above are cross-cut by a series of northeast striking diabase dikes. The weathered material is subdivided into saprolite and saprolite rock mix, regardless of parent material due to significant downslope movement. A non-mineralized metapelite unit is also modeled as an over thrust unit to the south.

Columbus has constructed a geologic model which includes the ten lithologies above. This has resulted in a detailed, 3-D geologic model created by using ARANZ Leapfrog® Geo software (Leapfrog®). These ten rock types constitute the basis of the block model.

14.3 Controls on Gold Mineralization

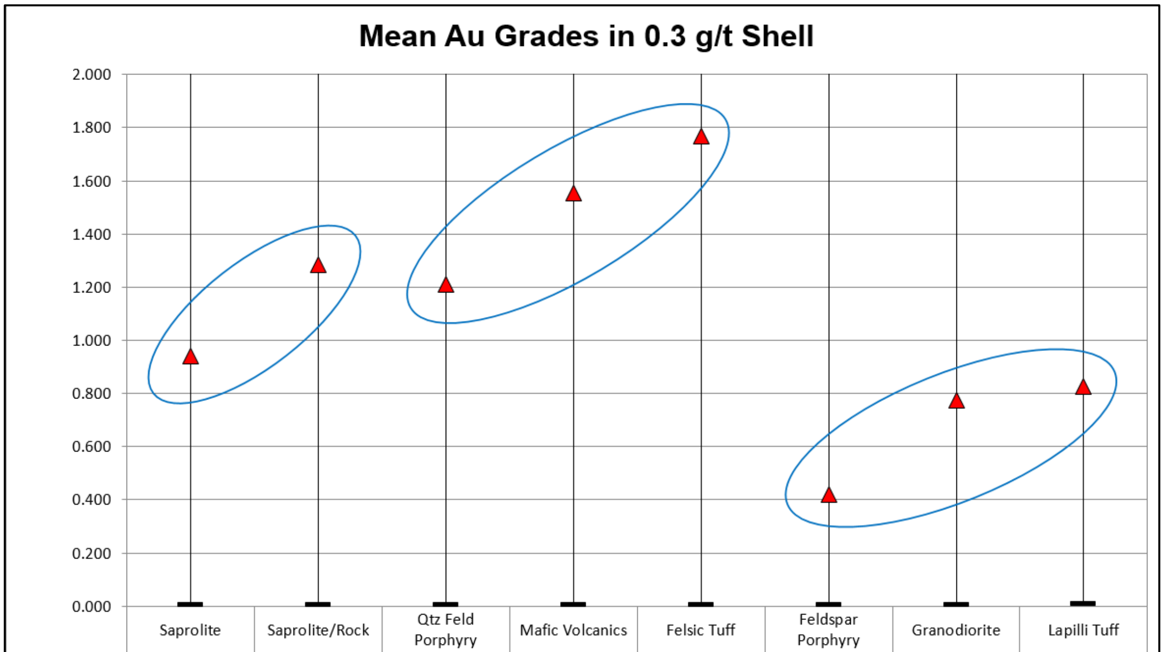
Gold mineralization is controlled mainly by structural fabric and lithology. The mineralization is localized in planar zones which have recurrent distribution and highly variable grades. Anomalous gold grades typically occur in zones 3 to 10 m wide which are separated by barren or lower grade zones 10 to 30 m wide. This is a common occurrence in these types of deposits and it is very important to consider this fact when designing reliable resource estimation. The orientation of this preferred plane of mineralization has been identified and refined over the past several years of drilling. Columbus has completed a program of oriented core drilling which has provided valuable information to better understand the structural geology of the deposit. All structural orientation data to date was acquired and plotted on lower hemisphere stereonets. The structural fabric data includes; foliation, shear planes, lithologic contacts and veins. The results of the stereonet plots are summarized in Table 14-1. These results confirm that the preferred orientation of mineralization as interpreted by Columbus, does follow along the average foliation and shear planes.

Table 14-1: Average Orientations of Structural Fabrics

Fabric	Strike	Dip °	# Measurements
Foliation	N86E	-70S	1,119
Shear Planes	N90E	-74S	35
Contacts	N83E	-70S	785
Veins	N87E	-71S	878

Source: SRK, 2016

To illustrate the importance of lithologic control of mineralization, SRK constructed a box plot of gold values hosted within a 0.3 g/t Au grade shell subdivided by lithology. The results are presented in Figure 14-1. The box plot shows three relative levels of mineralization controlled by lithology. Each of these three lithic groups were estimated as independent hard boundary domains.



	Saprolite	Saprolite/Rock	Qtz Feld Porphyry	Mafic Volcanics	Felsic Tuff	Feldspar Porphyry	Granodiorite	Lapilli Tuff
Q1	0.123	0.209	0.090	0.065	0.190	0.012	0.031	0.116
− Min	0.002	0.002	0.002	0.002	0.002	0.002	0.002	0.005
Median	0.362	0.528	0.270	0.440	0.542	0.074	0.118	0.350
▲ Mean	0.940	1.283	1.212	1.553	1.767	0.419	0.775	0.827
− Max	29.154	29.014	62.500	70.990	163.000	3.182	82.500	11.020
Q3	0.900	1.160	0.703	1.374	1.360	0.558	0.600	0.924
NSamples	1071	723	350	1282	8032	59	1540	152

Source: SRK, 2016

Figure 14-1: Box Plot of Gold Grade by Lithology

14.4 Density

Density testing was performed on the drill core during 2007 and from 2011 to 2016, a total of 3,607 density measurements were taken from all lithic varieties by onsite personnel. The averages of each lithology are listed in Table 14-2. These densities were assigned in the block model based on the lithology of the block.

Table 14-2: Densities Assigned in the Block Model

Rock Type	Number of Measurements	Minimum	Maximum	Average Density g/cm^3
Saprolite	412	1.17	2.67	1.736
Saprolite-Rock Transition	175	1.61	2.89	2.412
Felsic Tuff	1,319	1.55	4.53	2.893
Mafic Volcanics	442	2.73	4.33	3.131
Granodiorite	626	2.58	3.20	2.749
Feldspar Porphyry	73	2.62	2.87	2.777
Quartz-Feldspar Porphyry	117	2.66	3.10	2.779
Lapilli Tuff	80	2.63	3.17	2.827
Diabase Dikes	363	2.69	3.16	3.004
Total	**3,607**			

Source: SRK, 2016

14.5 Sample Database

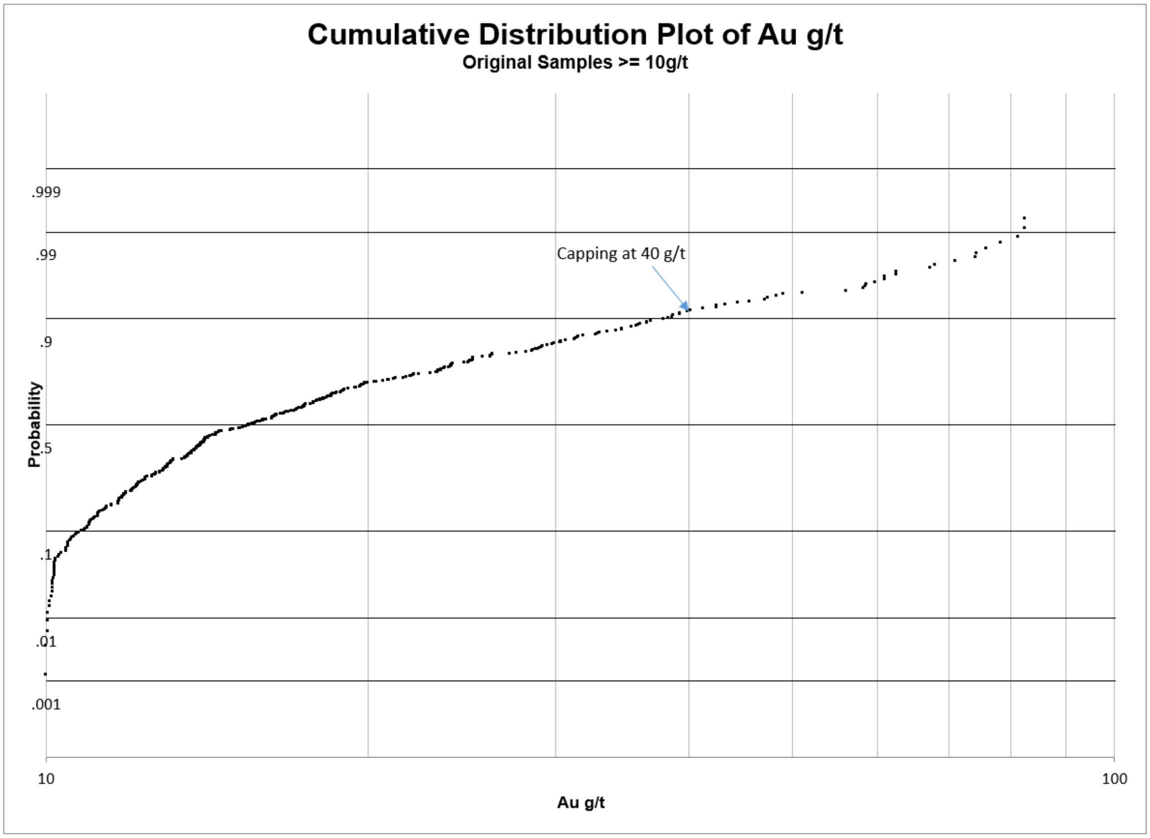
The July 1, 2016 database contains information from 349 diamond core and RC drillholes and 87 channel samples. The drilling was completed in two main campaigns. A previous owner drilled 56 holes between 1996 and 1998. Columbus completed an additional 293 holes from 2011 to March, 2016. The channel samples were all collected from surface between 1995 and 1997 and the majority are outside of the resource area. SRK has previously reviewed the 1995 through 1998 exploration data and found it to be of sufficient quality to support an industry standard, resource estimation.

The database includes four Excel® files containing information on collar locations, downhole surveys, lithology and gold assays. There are 59,862 valid entries in the assay file with an average sample length of 1.04 m.

14.6 Capping and Compositing

The original drillhole gold values were assessed for statistical outliers using a lognormal cumulative distribution plot and decile analysis. The decile analysis was used to identify the appropriate bin range for capping and the cumulative distribution plot was used to define the final capping level. The results of the cumulative distribution plot are presented in Figure 14-2. The Au capping level was chosen at 40 g/t mainly because this is the point where the cumulative distribution trends lose continuity and the data values above, show irregular distribution. The Au capping resulted in 31 samples ranging from 40.1 g/t to 163 g/t being reduced to 40 g/t prior to compositing. This was a net loss of 3.4% of all gold in the database.

Compositing was completed in 3 m downhole lengths with no breaks at lithologic contacts. The 3 m length was chosen as an appropriate size for two reasons. This length includes three original assay intervals so that it provides some smoothing of the data while still preserving the recurrent nature of the gold mineralization. The 3 m composite length also results in approximately two composites being included within the diagonal intersection of the 5 m, Y direction block size.



Source: SRK, 2016

Figure 14-2: Log Normal Cumulative Distribution Plot of Gold Assays above 10 g/t

14.7 Block Model

The block model limits of the SRK resource estimations are listed below. The block model coordinates are referenced to the "Official" French Guianian RGFG94 Zone 22N UTM coordinate system. The block dimensions are based on a compromise between the average drillhole spacing, a typical open pit selective mining unit, the variability of the mineralization and computational efficiency of keeping the model under ten million blocks. The block model limits and block sizes are listed in Table 14-3. There are 7,086,240 blocks in the model.

Table 14-3: Block Model Size and Extents

Orientation	Minimum (m)	Maximum (m)	Block Dimension (m)
Easting	172,200	175,160	10
Northing	520,300	521,250	5
Elevation (AMSL)	-150	480	5

Source: SRK, 2016

14.8 Estimation Strategy

Columbus constructed Leapfrog® software generated wireframe solids which enclose anomalous gold mineralization at a 0.3 g/t Au threshold. Numerous grade shells were constructed using a variety of sensitivities. The grade shells were evaluated for validity and functionality using three methods. First they were queried to determine what percentage of the available 3 m composite samples above 0.3 g/t Au are captured. The final grade shell captured 96% of all 3 m composite samples with Au grade above 0.3 g/t. Second, it was queried to determine how many samples within the grade shell fall above the 0.3 g/t threshold. In the final grade shell, 80% of the samples were above the threshold. Therefore, it has 20% dilution. Third, they were visually inspected to be sure the geometry was reasonable, based on the nearby drillholes.

Three rock groups were used as estimation domains as discussed in Section 14.3, Figure 14-1. Each rock group was estimated independently both internal and external to the grade shell using only samples from the same domain. The resultant grade estimation was therefore conducted in six domains. As discussed in Section 14.3, the gold mineralization is strongly controlled by thin planar zones. These generally strike east-west and dip approximately -68° south. Because the mineralization is extremely planar, a single search orientation along this plane was used for all bedrock lithologies. The saprolite and saprolite-rock lithologies were estimated using a search orientation parallel to the surface slope due to the significant amount of down slope movement documented in these lithologies. An IDW^2 algorithm was used for the grade estimations since the Au variograms have very high nugget values and short ranges.

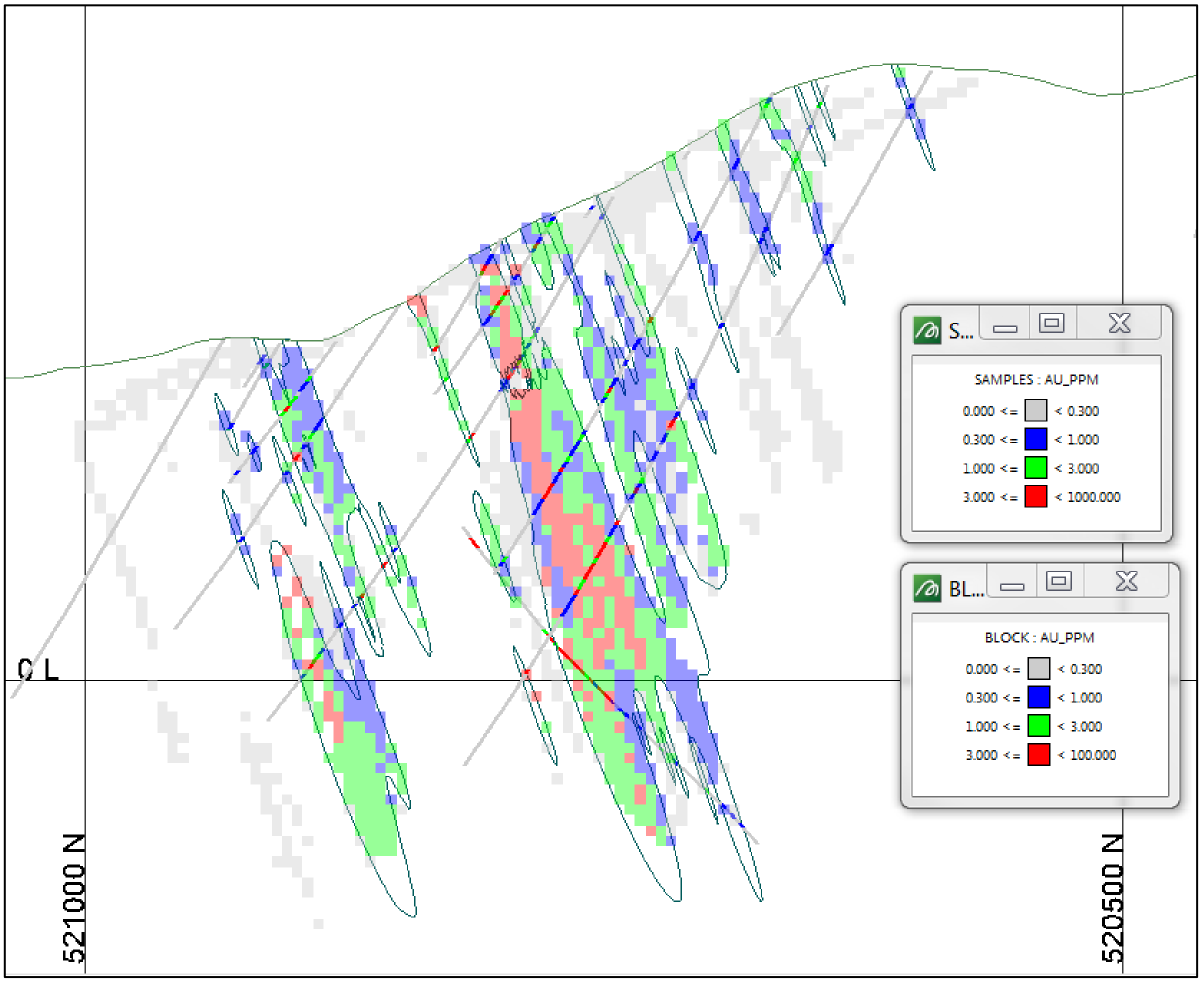
14.9 Estimations Procedures

The grade estimations for all metals in all domains within the Au grade shell, utilize a four pass sample search strategy with each pass searching longer distances than the previous. Outside of the grade shell, a three pass sample search strategy was used. In all domains, only blocks located within 75 m to the closest sample were included as the final estimation. Because the grade shell and distance restriction has been predetermined; and mineralized blocks are now isolated from less-mineralized blocks, the model is allowed to search relatively longer distances in the preferred plane of mineralization and shorter distance in the direction normal to it. This method provides for a larger pool of composites to be considered resulting in appropriate grade smoothing. The search distances and sample selection criteria are listed in Table 14-4. Sample length weighting is used in all estimations to account for any short composites located at the ends of drillholes. As part of the grade estimation, model validation is conducted as an interactive process. To achieve proper validation, some higher grade composites were limited by the distance they could be interpolated. A high-grade composite restriction, as listed in Table 14-4, means that any sample above the listed grade could only be interpolated over the listed distance. Figure 14-3 and Figure 14-4 show representative cross sections of the gold estimation results.

Table 14-4: Au Grade Estimation Parameters

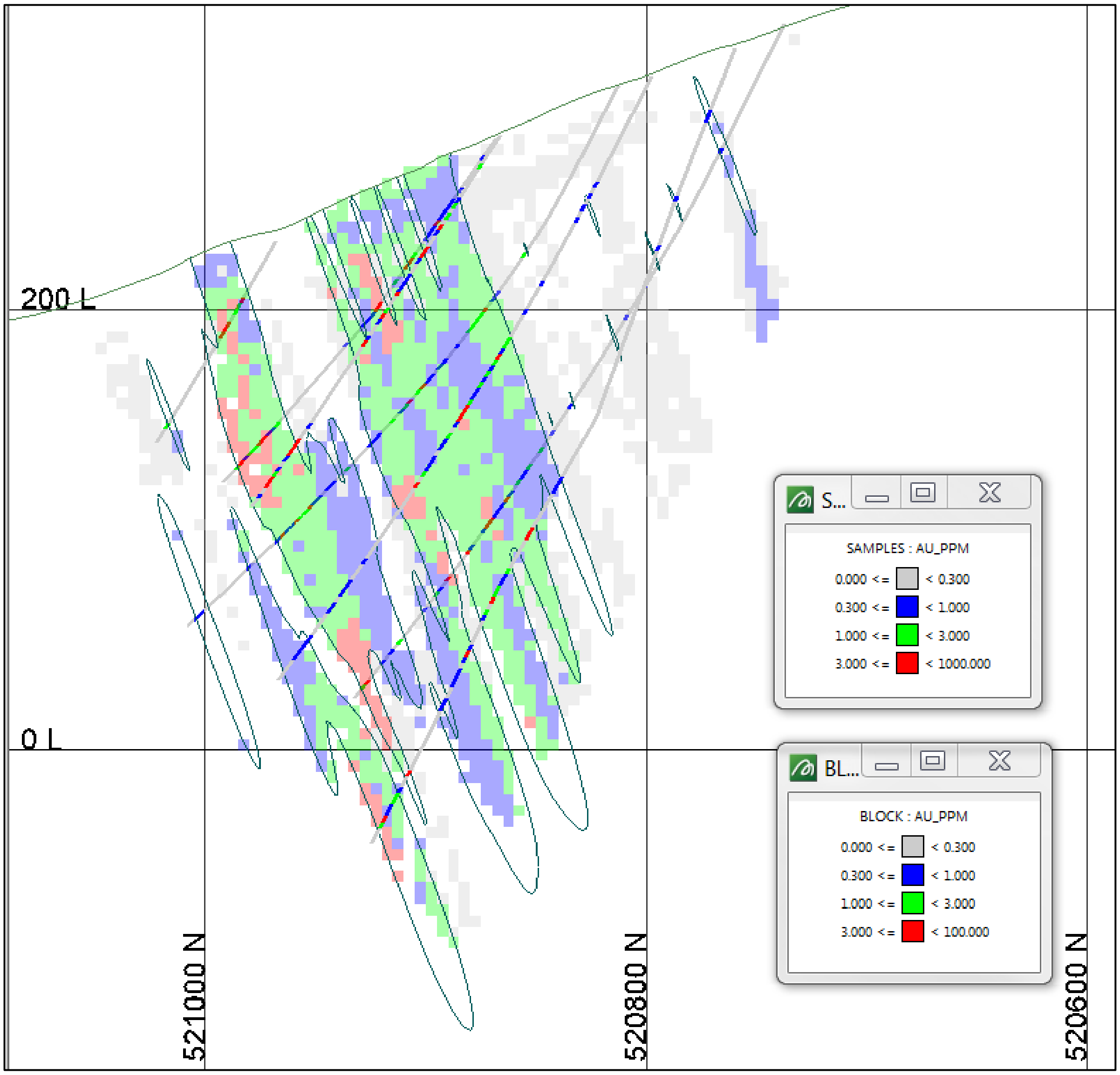
Estimation	Estimation Pass	Search Range (x,y,z) m	Min/Max Samples	Octant Restriction	High Grade Composite Restriction (grade. X, y, z distances)
Saprolite/Sap Rock Inside Grade Shell	1	5,2.5,2.5 (Box)	1/3	None	None
	2	35,35,5	3/8	2 Samp/Oct	
	3	65,65,10	3/8	2 Samp/Oct	
	4	125,125,15	3/8	2 Samp/Oct	
Saprolite/Sap Rock Outside Grade Shell	1	35,35,5	3/8	2 Samp/Oct	None
	2	65,65,10	3/8	2 Samp/Oct	
	3	125,125,15	3/8	2 Samp/Oct	
Felsic Tuff, Mafic Volcanics, QFP Inside Grade Shell	1	5,2.5,2.5 (Box)	1/3	None	None
	2	35,35,5	3/8	2 Samp/Oct	None
	3	65,65,10	3/8	2 Samp/Oc	>12 g/t <50 m, 50 m, 5 m
	4	125,125,15	3/8	2 Samp/Oct	>12 g/t <50 m, 50 m, 5 m
Felsic Tuff, Mafic Volcanics, QFP Outside Grade Shell	1	35,35,5	3/8	2 Samp/Oct	None
	2	65,65,10	3/8	2 Samp/Oct	
	3	125,125,15	3/8	2 Samp/Oct	
All Other Lithologies Inside Grade Shell	1	5,2.5,2.5 (Box)	1/3	None	None
	2	35,35,5	3/8	2 Samp/Oct	None
	3	65,65,10	3/8	2 Samp/Oct	>5 g/t <50 m, 50 m, 5 m
	4	125,125,15	3/8	2 Samp/Oct	>5 g/t <50 m, 50 m, 5 m
All Other Lithologies Outside Grade Shell	1	35,35,5	3/8	2 Samp/Oct	None
	2	65,65,10	3/8	2 Samp/Oct	
	3	125,125,15	3/8	2 Samp/Oct	

Source: SRK, 2016



Source: SRK, 2016

Figure 14-3: Representative Cross Section 173,000 E with Estimated Au Grades (Viewing East)



Source: SRK, 2016

Figure 14-4: Representative Cross Section 174,000E with Estimated Au Grades (Viewing East)

14.10 Model Validation

Six techniques were used to evaluate the validity of the block model. First, the interpolated block grades were visually checked on sections, plan views and in 3-D for comparison to the composite assay grades. Second, the general model estimation parameters were reviewed to evaluate the performance of the model with respect to supporting data. This included the number of composites used, number of drillholes used, average distance to samples used, and the number of blocks estimated in each pass. The results of this analysis are presented in Table 14-5. These show that the blocks are well informed from sufficient samples, selected from multiple drillholes at a reasonable distance. Third, statistical analyses were made comparing the estimated block grades from the IDW^2 estimation to the composite sample data supporting the estimation. Table 14-6 lists the results of the statistical comparison. In all cases, the block grades are very close to, or slightly below, the composite grades as desired. Fourth, a nearest neighbor estimation was run using a single composite to estimate each block using the same parameters as the IDW^2 estimation. The total contained metal, at a zero CoG in the nearest neighbor estimation, is compared to the IDW^2 estimation at the same cut-off. The results of this comparison are listed in Table 14-7. The nearest neighbor estimation shows a small discrepancy in the Saprolite/Saprolite Rock unit however all of the other units have less metal in the IDW^2 estimation. The fifth validation was to construct N-S oriented swath plots located every 50 m spacing. The results shown in Figure 14-5 illustrate strong correlation between block grades and composites with an appropriate amount of smoothing. The final validation was an assessment of the impacts of edge dilution about the margins of the Au grade shell. To quantify the impacts of dilution, a partial Au grade estimation was completed for all blocks touching the grade shell. These blocks were estimated with samples internal to the wireframe and then again with the samples external to the wireframe and a final diluted Au grade was calculated based on each of the grade estimations weighted by the proportion of the block representing the estimation. The results of the diluted model are compared to the undiluted model in Table 14-8. This shows that as the CoG increases, there is a net loss in contained metal.

Table 14-5: Estimation Performance Parameters of Au Estimation in Grade Shell

Estimation	Samples Used (#)	Drillholes Used (#)	Average Distance to Samples (m)	Blocks Estimated (%)
Pass 1	1.3	1	2.7	2
Pass 2	3.8	2.3	23	11
Pass 3	5.1	2.9	39	55
Pass 4	6.2	3.8	72	32
All Passes	5.2	3.1	47	100

Source: SRK, 2016

Table 14-6: Model Validation Statistical Results in Grade Shell

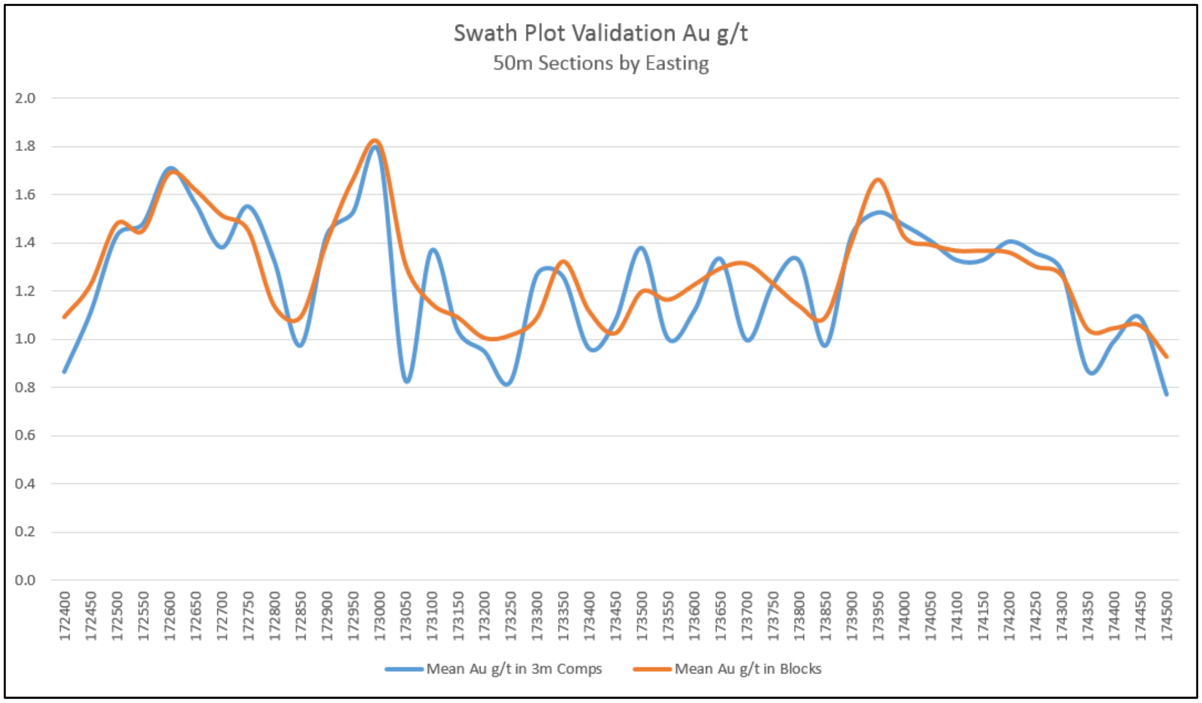
Estimation	Average Composite Grade (g/t)	Average Block Grade (g/t)	Difference of Composites to Blocks (%)
Saprolite/Sap Rock	1.079	1.035	4.1
Felsic Tuff/Mafic Volcanics	1.420	1.380	2.8
All Other Lithologies	1.007	0.970	3.6
All Lithologies Combined	1.3018	1.2958	0.5

Source: SRK, 2016

Table 14-7: Model Validation nearest Neighbor Results in Grade Shell

Estimation	Cut-off (g/t)	Tonnes (M)	IDW^2 Grade (g/t)	NN Au Grade (g/t)	% Difference of Metal Mass, IDW^2 to NN
Saprolite/Sap Rock	0	9.3	1.0381	1.3024	3.8
Felsic Tuff/Mafic Volcanics	0	99.7	1.3788	1.3768	0.2
All Other Lithologies	0	13.5	0.9702	0.9635	0.7
All Lithologies Combined	0	122.6	1.3078	1.3024	0.4

Source: SRK, 2016



Source: SRK, 2016

Figure 14-5: North-South Oriented Swath Plots

Table 14-8: Model Validation Statistical Results in Grade Shell

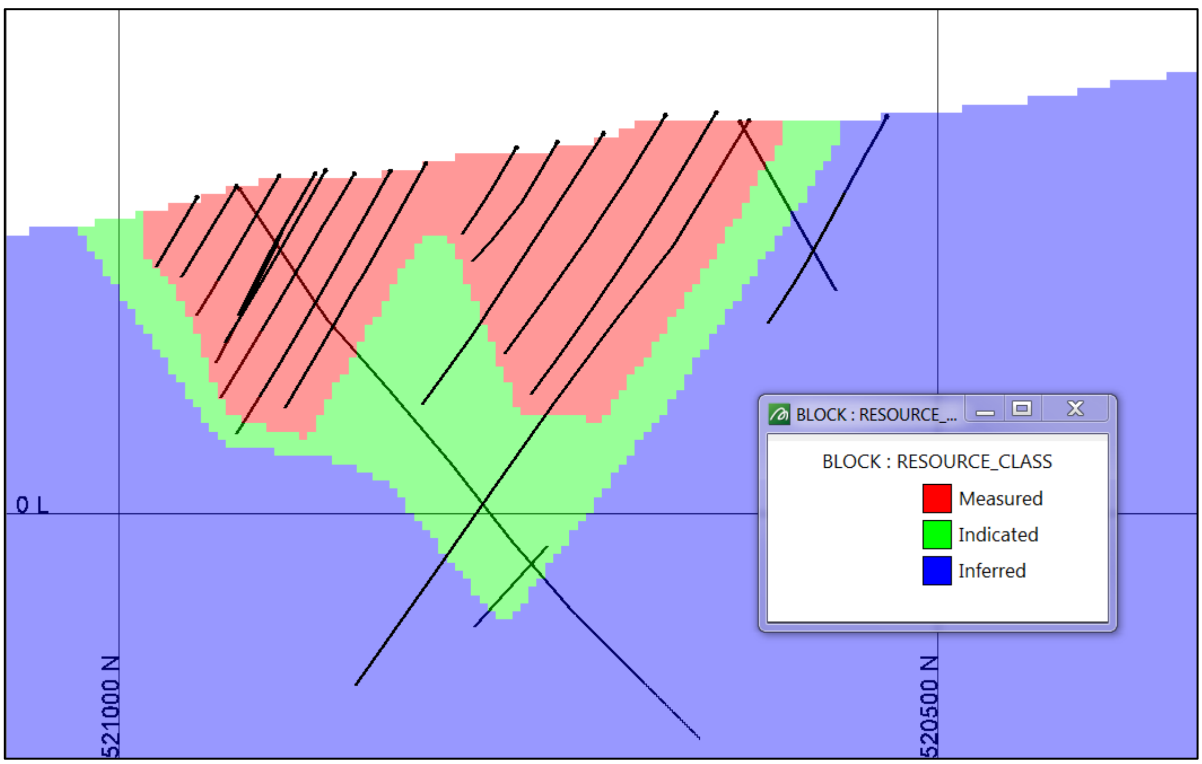
Au g/t	Undiluted Model			Diluted Model			Percentage Difference		
Cut-off	Au g/t	Tonnes (M)	Ounces (M)	Au g/t	Tonnes (M)	Ounces (M)	Grade	Tonnes	Ounces
0.1	0.8122	215.0	5.6	0.707	248.8	5.7	-13.0	15.7	0.7
0.2	1.2512	130.5	5.2	0.983	167.5	5.3	-21.4	28.4	0.9
0.3	1.3293	121.2	5.2	1.120	141.3	5.1	-15.7	16.6	-1.8
0.4	1.4004	112.9	5.1	1.248	121.1	4.9	-10.9	7.2	-4.5
0.5	1.5021	102.0	4.9	1.378	104.1	4.6	-8.3	2.1	-6.3
0.6	1.6137	91.3	4.7	1.506	90.2	4.4	-6.7	-1.2	-7.8
0.7	1.7313	81.4	4.5	1.635	78.4	4.1	-5.6	-3.6	-9.0

Source: SRK 2016

14.11 Resource Classification

Mineral Resources are classified under the categories of Measured, Indicated and Inferred according to CIM guidelines. Classification of the Mineral Resources reflects the relative confidence of the grade estimates and the continuity of the mineralization. This classification is based on several factors including sample spacing relative to geological and geo-statistical observations regarding the continuity of mineralization, data verification to original sources, specific gravity determinations, accuracy of drill collar locations, accuracy of topographic data, quality of the assay data and many other factors which influence the confidence of the mineral estimation. No single factor controls the Mineral Resource classification, rather each factor influences the end result.

The Mineral Resources reported for the Montagne d'Or deposit are classified as Measured, Indicated and Inferred Mineral Resources, based primarily on drillhole spacing since all other supporting data is of good quality. A wire frame solid was constructed around the area where the average drillhole spacing is approximately 35 m or less and these were used to assign the Measured Mineral Resource classification. This is a focused area of drilling completed in 2015 and 2016 located within the proposed Phase I pit. The measured wire frame solid is flanked by a second wireframe constructed around the areas where the average drillhole spacing is approximately 65 m or less and these were used to assign the Indicated Mineral Resource classification. All blocks outside of these wireframes were classified as Inferred Mineral Resources. Figure 14-6 presents a representative cross section (172,600 E) showing the resource classification.



Source: SRK, 2016

Figure 14-6: Representative Cross Section 172,600 E Showing Resource Classification (Viewing East)

14.12 Mineral Resource Statement

The Montagne d'Or Mineral Resource Statement is presented in Table 14-9. The resource is confined within a Whittle™ optimization pit shell and a CoG of 0.4 g/t Au applied. The pit shell and CoG assumes open-pit mining methods and is based on a mining cost of US$2/t, milling cost of US$15/t, administration cost of US$1/t, a gold price of US$1,300/oz., 95% gold recovery, gold refining cost of US$8/oz, and 5% NSR royalty. A 45° pit shell slope was used for bedrock and a 35° pit shell slope was used for saprolite. The reported Mineral Resources include material from all estimation domains.

The effective date for the Mineral Resource estimate in this report is July 1, 2016 and was prepared by SRK. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Table 14-9: Montagne d'Or Mineral Resource Statement as of July 1, 2016 SRK Consulting (U.S.), Inc.

Classification	Au Cut-Off (g/t)	Tonnes (M)	Au (g/t)	Contained Au (Moz)
Measured	0.4	10.3	1.804	0.60
Indicated	0.4	74.8	1.350	3.25
M & I	0.4	85.1	1.405	3.85
Inferred	0.4	20.2	1.484	0.96

- All figures rounded to reflect the relative accuracy of the estimates.
- Metal assays were capped where appropriate.
- The Mineral Resources were estimated by Bart A. Stryhas PhD, CPG # 11034, a Qualified Person.
- Mineral Resources are reported based on a CoG of 0.4 g/t Au, and are reported inside a conceptual pit shell based on appropriate mining and processing costs and metal recoveries for oxide and sulphide material.
- CoGs are based on a mining cost of US$2/t, milling cost of US$15/t, administration cost of US$1/t, a gold price of US$1,300/oz., 95% gold recovery, gold refining cost of US$8/oz, and 5% NSR royalty.
- Silver was not included in the resource estimate. No gold equivalent grades are reported.

Source: SRK, 2016

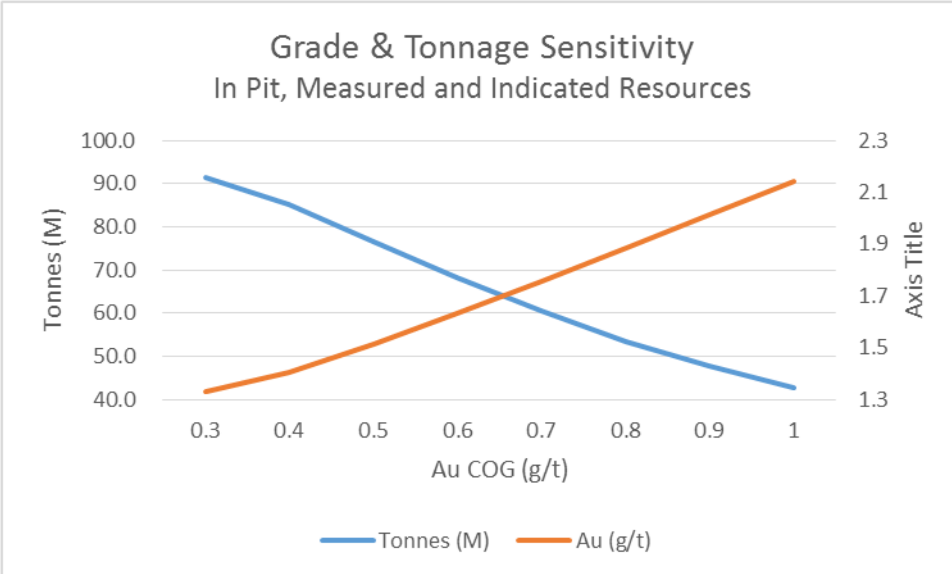
14.13 Mineral Resource Sensitivity

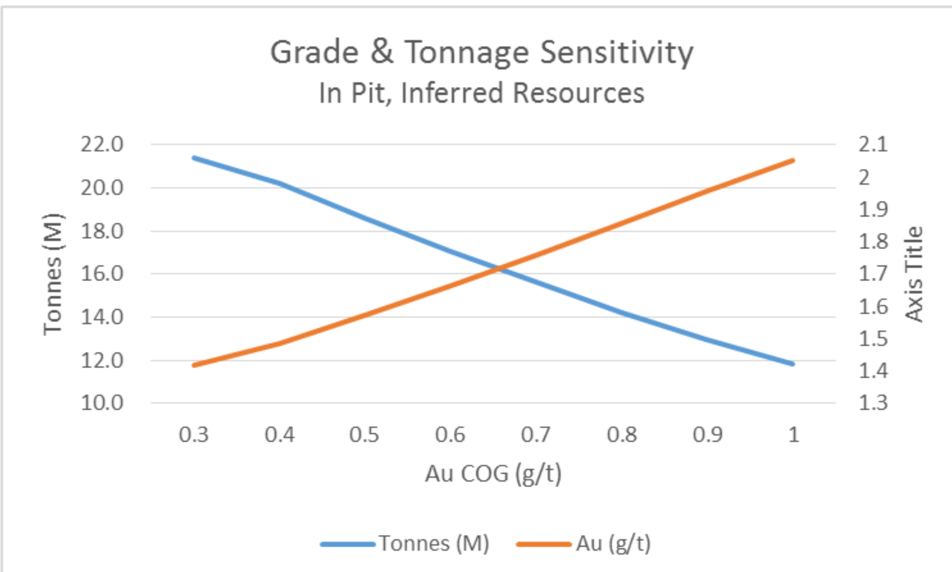
The Mineral Resources shown in Table 14-10 are presented at a range of CoGs, subdivided by resource classification. Graphical representations of the grade and tonnage sensitivities of the Indicated resources are presented in Figure 14-7. All resources are confined within the Whittle™ optimization pit shell.

Table 14-10: Mineral Resource Sensitivity [1]

Measured and Indicated			
Cut-off	**Tonnes (M)**	**Au (g/t)**	**Au (Moz)**
0.3	91.5	1.332	3.92
0.4 [2]	85.1	1.405	3.85
0.5	76.6	1.511	3.72
0.6	68.1	1.631	3.57
0.7	60.4	1.757	3.41
0.8	53.5	1.886	3.24
0.9	47.7	2.014	3.09
1.0	42.6	2.141	2.93
Inferred			
Cut-off	**Tonnes (M)**	**Au (g/t)**	**Au (Moz)**
0.3	21.4	1.42	0.98
0.4 [2]	20.2	1.484	0.96
0.5	18.6	1.571	0.94
0.6	17.1	1.664	0.91
0.7	15.6	1.758	0.88
0.8	14.2	1.856	0.85
0.9	12.9	1.957	0.81
1.0	11.8	2.052	0.78

(1) Tonnes and grade have been rounded to reflect the level of expected accuracy.
(2) Base Case CoG.
Source: SRK, 2016





Source: SRK, 2016

Figure 14-7: Sensitivity of Tonnes and Grade to Cut-off

14.14 Mineral Reserve Block Model Estimate

The block model supporting the Mineral Reserve was constructed and estimated in the identical manor as the Mineral Resource model with the following exceptions. The Mineral Reserve block model utilized a regular 5 m x 5 m x 5 m, x,y,z block size respectively. The grade of each block which intercepted the 0.3 g/t Au grade shell was estimated twice; first using the samples internal to the grade shell and second using the samples external to the grade shell. The final grades of these blocks were calculated using the weighted proportions of the block internal and external to the grade shell against the respective estimated grades. All reserve pit design and scheduling is based on this block model.

14.15 NAG-PAG Block Model Estimate

A block model was also created to estimate the NAG and PAG of the proposed mining plan.

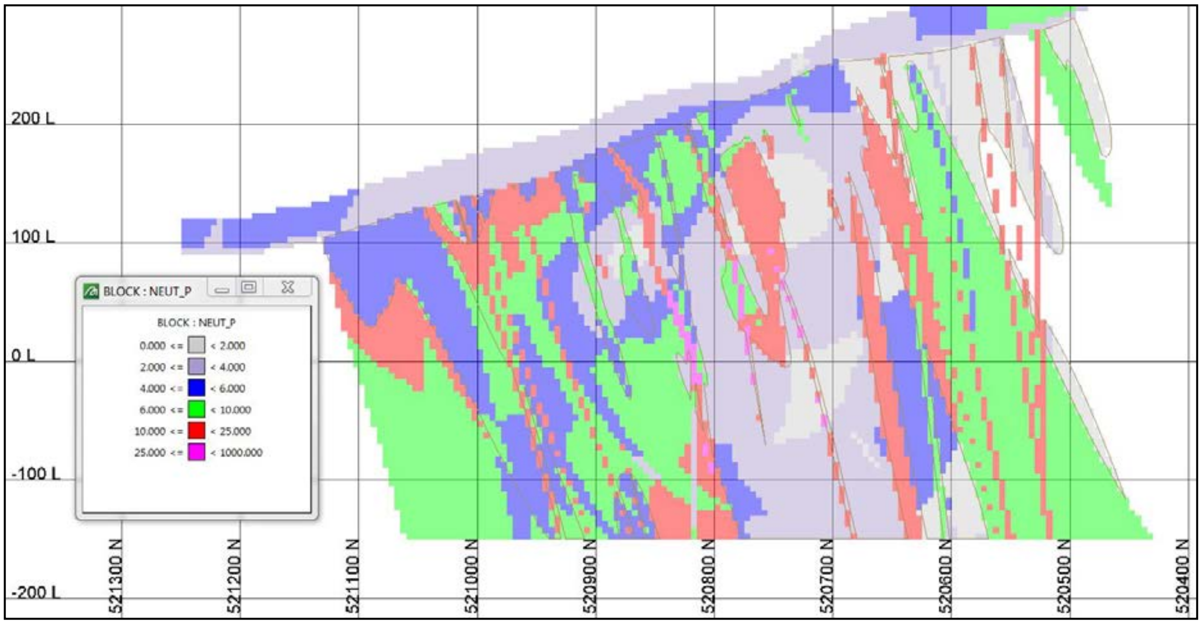
14.15.1 Database and Block Model

The database supporting the NAG-PAG block model includes 449 static test samples. The test work is described in detail in SRK (2017a) but essentially includes the Acid Neutralization Potential (ANP) and the AGP values for a variety of lithologic types located throughout the potential pit area. The ANP data correlates to the NAG values and the AGP data correlates to the PAG values. Table 14-11 list the number of static test samples by lithology.

The reserve block model was used to generate the NAG-PAG model since it is used primarily in the mining plan. Details of the block model are presented in Section 14.14. Each block is 5 m x 5 m by 5 m in the x, y, and z directions respectively. Each model block is assigned one of nine lithologic types according to the geologic model supporting the resource estimation.

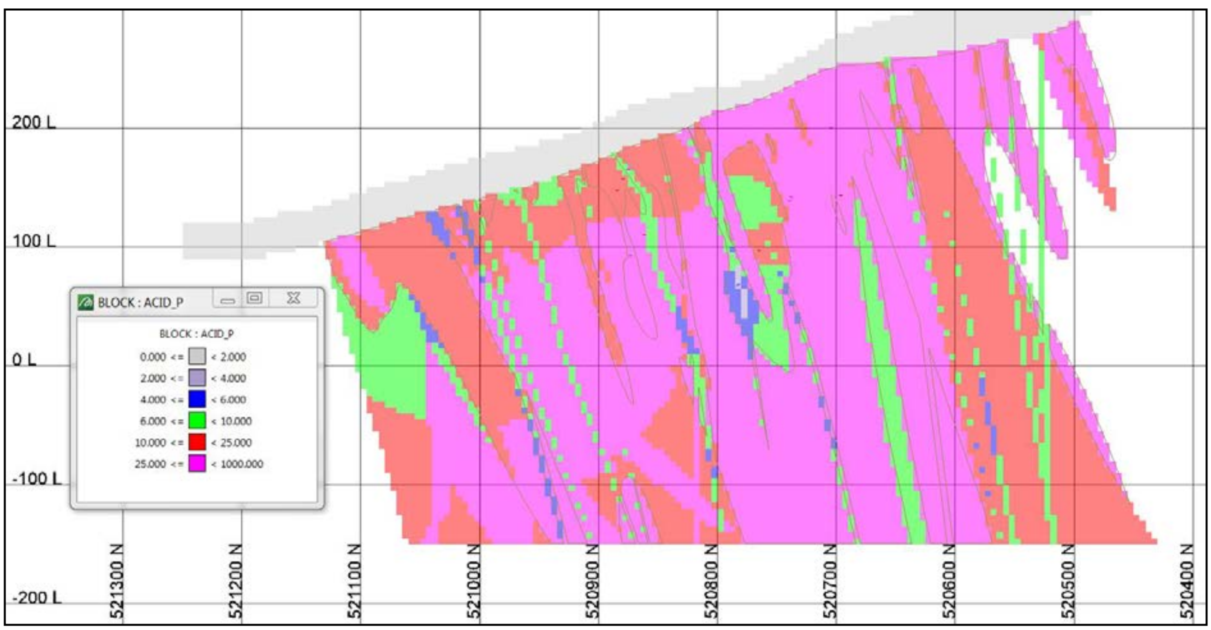
14.15.2 Estimation Procedures

Each of the nine lithologies are considered as independent domains for the assignment and estimation of their NAG and PAG values. The ANP and AGP data values were sorted by lithology and plotted on cumulative frequency distribution graphs to assess hi and low outlier values. For each lithology, outlier values were identified and excluded from the main population for the calculation of average ANP and AGP values of that unit. The results of the outlier assessment and averages are reported in Table 14-11. For each lithology in the block model, the average ANP and AGP values were assigned to establish a background value. The actual estimation of ANP and AGP was then conducted over the background values. The database supporting the estimation included the filtered data values with all hi and low outliers removed. The estimation utilized a three pass search strategy with each sequential pass searching longer distances. The search ellipsoids are based on the fabric of the lithologies so that the estimation will try to search within each rock type. Due to the relatively low population of data in some units, no octant or drillhole restrictions were applied. The general estimation parameters are listed in Table 14-12. An Inverse Distance Weighting Squared (IDW^2) algorithm was used to estimate grade. This was chosen to improve the statistical block validation to sample data. Representative cross sections of the NAG and PAG estimation are presented in Figure 14-8 and Figure 14-9 respectively. Once the final NAG and PAG were estimated the NPR was calculated as NPR=NAG/PAG. This was used to partially support the geochemical analysis of the potential WRDs and tailings.



Source: SRK 2016

Figure 14-8: Block Model Estimated NAG (Section 173,200E Viewing East)



Source: SRK 2016

Figure 14-9: Block Model Estimated PAG (Section 173,200E Viewing East)

14.15.3 Model Validation

Three techniques were used to evaluate the validity of the block model. First, the interpolated block grades were visually checked on sections, plan views and in 3-D for comparison to the support samples. Second, the general model estimation parameters were reviewed to evaluate the performance of the model with respect to the supporting data. This included the number of composites used and the number of block estimated in each pass. The results of this analysis are presented in Table 14-13. These show that the estimated blocks are reasonably well informed from sufficient samples. Third, statistical analyses were made comparing the estimated block values from the IDW^2 estimation to the composite sample data supporting the estimation. Table 14-14 lists the results of the statistical comparison. In most cases, the block values are reasonably close to the samples.

The resulting model is a high level prediction of the NAG and PAG based on the relative sampling density of only 449 data points. These results are used to support the geochemical model of the mining plan.

Table 14-11: AGP & ANP Outlier Identification and Averaging by Lithology

Data Set	Lithology	Data Range Utilized in Model	Number of Samples Included	Average Value of Utilized Data	Excluded, Outlier Data Values
AGP	Saprolite	All	74	0.2	None
	Saprolite Rock	0-1.0	11	0.2	1.8,14.6,60.7
	Felsic Tuff	3-200	128	47.2	0.5,1.8,222,243
	Feldspar Porphyry	0-30	11	10.0	77
	Granodiorite	1-55	83	18.0	0.2,0.3,0.5,84,93,107,115
	Lapilli Tuff	All	12	78.1	
	Quartz Feldspar Porphyry	All	11	37.0	
	Mafic Volcanics	2.5-125	38	29.3	0.2,0.3,0.6,0.9,1.2,1.5,188,256
	Diabase Dikes	4-15	39	8.3	2.7,3.4,3.6,20,26,36,48,59,82
ANP	Saprolite	1-22	70	7.1	0.4,0.8,31,63
	Saprolite Rock	1-8	15	3.7	0.4
	Felsic Tuff	0-16	129	3.1	22,29
	Feldspar Porphyry	10-43	9	25.6	2.5.3.3,6.7
	Granodiorite	1.5-27	80	9.6	0.4,0.8,76
	Lapilli Tuff	All	12	9.2	
	Quartz Feldspar Porphyry	All	11	11.9	
	Mafic Volcanics	1-38	42	11.9	0.4,73,80,87
	Diabase Dikes	1-39	46	13.8	0.8,130

Source: SRK 2016

Table 14-12: General Estimation Parameters

Lithology	Estimation Pass	Min/Max Number of Samples	Search Orientation (bearing, plunge, dip)	Search Distance (x, y, z)(m)
Saprolite and Saprolite Rock	1 2 3	3/8	0, -25, 0	100, 100, 50 150, 150, 75 250, 250, 125
All Bedrock Lithologies	1 2 3	3/8	174, -70, 0	100, 100, 50 150, 150, 75 250, 250, 125

Source: SRK 2016

Table 14-13: Estimation Parameter Results

Lithology	Estimation Pass	Number of Samples Used	Percentage of Total Blocks Estimated
Saprolite	1	4.7	20
	2	4.5	28
	3	5.5	52
	All	5.1	100
Saprolite Rock	1	3	1
	2	3.1	11
	3	3.5	88
	All	3.5	100
Felsic Tuff	1	4.4	19
	2	4.9	28
	3	6	53
	All	5.4	100
Feldspar Porphyry	1	3	5
	2	3.1	15
	3	3.2	80
	All	3.2	100
Granodiorite	1	4.9	13
	2	4.6	22
	3	5.7	65
	All	5.4	100
Lapilli Tuff	1	3	1
	2	3.3	22
	3	4.6	77
	All	4.3	100
Quartz Feldspar Porphyry	1	3	2
	2	3.1	13
	3	3.4	85
	All	3.3	100
Mafic Volcanics	1	4.3	14
	2	4.8	28
	3	5.8	58
	All	5.3	100
Diabase Dikes	1	3.4	6
	2	3.9	21
	3	4.9	73
	All	4.6	100

Source: SRK 2016

Table 14-14: Statistical Validation Results

Lithology	Sample AGP	Block PAG	% Diff Block to Sample
Saprolite	0.176	0.181	3
Saprolite Rock	0.155	0.162	4
Felsic Tuff	47.154	51.381	9
Feld Porphyry	10.035	7.555	-25
Granodiorite	17.980	18.606	3
Lapilli	78.140	81.522	4
QFP	37.010	43.013	16
Mafic V	29.270	25.796	-12
Diabase	8.310	7.635	-8
Lithology	**Sample ANP**	**Block NAG**	**% Diff Block to Sample**
Saprolite	7.065	6.969	-1
Saprolite Rock	3.722	3.373	-9
Felsic Tuff	3.062	2.673	-13
Feld Porphyry	25.555	27.893	9
Granodiorite	9.625	9.745	1
Lapilli	9.236	7.999	-13
QFP	11.894	10.398	-13
Mafic V	11.924	11.999	1
Diabase	13.804	11.942	-13

Source: SRK 2016

14.16 Relevant Factors

There are no additional relevant factors that are material to the current resource estimate. All resources are stated as in situ, and no modifying factor for mining dilution or mining recovery have been applied.

15 Mineral Reserve Estimate

LoM plans and resulting Mineral Reserves are determined based on a gold price of US$1,200/oz Au for the Montagne d'Or Project. Reserves stated in this report are dated effective as of September 1, 2016 with an EURUSD of US$1.10:€1.00.

The ore material is converted from Mineral Resource to Mineral Reserve based primarily on positive cash flow pit optimization results, pit design and geological classification of Measured and Indicated resources. The in situ value is derived from the estimated grade and certain modifying factors.

The Qualified Person has not identified any risk including legal, political or environmental, that would materially affect potential development of the Mineral Reserves, as of September 1, 2016.

15.1 Conversion Assumptions, Parameters and Methods

The conversion of Mineral Resource to Mineral Reserve entails the evaluation of modifying factors that should be considered in stating a Mineral Reserve. Table 15-1 illustrates a reserve checklist and associated commentary on the risk factors involved for the Montagne d'Or Mineral Reserve statement.

Table 15-1: Montagne d'Or SRK Reserve Checklist

Unit	Data Evaluated	Data Not Evaluated	Not Applicable	Notes
Mining				Moderate sized equipment fleet, pioneering support, sized for SMU of reserve
Mining Width	X			40 m+ where possible
Open Pit and/or Underground	X			Open pit
Density and Bulk handling	X			Hard Rock/Saprolite breakdown
Dilution	X			Mine model with SMU of 5 m x 5 m x 5 m limited to grade shell
Mine Recovery	X			Full mine recovery assumed
Waste Rock	X			No storage limitation, ARDML potential
Grade Control	X			Assume SMU and drill pattern
Processing	X			Full metallurgical test work program
Representative Sample	X			Lithological types, spatial representation and grade is representative
Product Recoveries	X			Full metallurgical test work
Hardness (Grindability)	X			Metallurgical test work
Bulk Density	X			Geotechnical studies
Deleterious Elements	X			None identified for process type, Hard/Saprolite blend considered
Process Selection	X			CIL
Geotechnical/Hydrological	X			
Slope Stability (Open Pit)	X			Geotechnical stability analysis FOS > 1
Water Balance	X			Full water balance study
Area Hydrology	X			Tropical environment, Pit closure study
Seismic Risk	X			Geotechnical study
Environmental				
Baseline Studies	X			WSP, EIS
Tailing Management	X			Adequate capacity
Waste Rock Management	X			Design close to pit rims no limit to vertical expansion
Acid Rock Drainage Issues	X			Active management required
Closure and Reclamation Plan	X			Closure cost estimated
Permitting Schedule	X			TBD, French jurisdiction
Location and Infrastructure				
Climate	X			Tropical, seasonal wet/dry seasons
Supply Logistics	X			Import by sea to FG and off-highway for 120km
Power Source(S)	X			Powerline to national grid
Existing Infrastructure	X			Poor
Labour Supply and Skill Level	X			French expatriates, training required
Marketing Elements or Factors				
Product Specification and Demand	X			Gold market
Off-site Treatment Terms and Costs	X			Gold market
Transportation Costs	X			Gold market
Legal Elements or Factors	X			Permitting ongoing
Security of Tenure	X			Permitting in progress, mineral rights in possession
Ownership Rights and Interests	X			Actively managed
Environmental Liability	X			Illegal mining history
Political Risk (e.g., land claims, sovereign risk)	X			French Government, NGOs of possible concern
Negotiated Fiscal Regime	X			EU country
General Costs and Revenue Elements or Factors				
General and Administrative Costs	X			Feasibility study level estimate
Commodity Price Forecasts	X			Company and street estimate
Foreign Exchange Forecasts	X			At time of BFS estimate
Inflation	X			Euro related
Royalty Commitments	X			Included in BFS
Taxes	X			Included in BFS
Corporative Investment Criteria	X			Nordgold/Columbus
Social Issues				
Sustainable Development Strategy	X			Environmental/Social operating plan
Impact Assessment and Mitigation	X			Environmental/Social operating plan
Negotiated Cost/Benefit Agreement	X			Environmental/Social operating plan
Cultural and Social Influences	X			Environmental/Social operating plan/NGO risk

Source: SRK, 2016

15.2 Reserve Estimate

LoM plans and resulting Mineral Reserves are determined based on a gold price of US$1,200/oz Au. Reserves stated in Table 15-2 are dated effective as of September 1, 2016 with an EURUSD of US$1.10:€1.00.

The ore material is converted from Mineral Resource to Mineral Reserve based primarily on positive cash flow pit optimization results, pit design and geological classification of Measured and Indicated resources. The in situ value is derived from the estimated grade and certain modifying factors.

The Qualified Person has not identified any risk including legal, political or environmental, that would materially affect potential development of the Mineral Reserves, as of September 1, 2016.

Table 15-2: Montagne d'Or Mineral Reserve Estimate as of September 1, 2016, SRK Consulting (U.S.), Inc.

Class	Tonnes M	Au g/t	Contained Au Moz
Proven	8.25	1.99	0.53
Probable	45.87	1.50	2.22
Proven and Probable	**54.11**	**1.58**	**2.75**

- Mineral Reserves are reported at varied cut-offs dependent on lithological rock types, economics and estimated metallurgical recovery. Felsic Tuffs have CoG of 0.617 g/t Au, Granodiorites have a CoG of 0.622 g/t Au, Mafics have a CoG of 0.665 g/t Au, Saprolite and Saprock have a CoG of 0.552 g/t Au.
- Associated metallurgical recoveries have been estimated as 93.8% for Felsic Tuffs, 95.2% for Granodiorites, 91.3% for Mafics and 96.4% Saprolite/Saprock
- Full mining recovery assumed.
- Reserves have no additional dilution added to that that inherent in the SMU of 5 m x 5 m x 5 m diluted mine block model.
- Reserves are based on a US$1,200/oz Au gold price.
- Reserves are converted from resources through the process of pit optimization, pit design, production schedule and supported by a positive cash flow model.
- The ore reserves were estimated by Bret C Swanson, BE (Min) MMSAQP #04418QP, a Qualified Person.
- Silver was not included in the reserve estimate. No gold equivalent grades are reported.
- The reserves are valid as of September 1, 2016.

Table 15-3: Montagne d'Or Rock Type Reserve Estimate as of September 1, 2016, SRK Consulting (U.S.), Inc.

Class	Rock Type	Tonnes M	Au g/t	Contained Au Moz
Proven	flpy	0.09	1.02	0.00
	fltf	5.92	2.21	0.42
	gran	0.69	1.31	0.03
	lptf	0.00	0.75	0.00
	mfvl	0.10	1.64	0.01
	qtzp	0.11	1.82	0.01
	sap	0.88	1.38	0.04
	sapr	0.45	1.58	0.02
	Sub Total Proven	***8.25***	***1.99***	***0.53***
Probable	flpy	0.34	1.37	0.02
	fltf	32.19	1.55	1.61
	gran	2.30	1.31	0.10
	lptf	0.10	0.86	0.00
	mfvl	5.86	1.64	0.31
	qtzp	0.23	1.20	0.01
	sap	3.06	1.20	0.12
	sapr	1.80	1.10	0.06
	Sub Total Probable	***45.87***	***1.50***	***2.22***
Total Proven and Probable	**All**	**54.11**	**1.58**	**2.75**

- Mineral Reserves are reported at varied cut-offs dependent on lithological rock types, economics and estimated metallurgical recovery. Felsic Tuffs have CoG of 0.617 g/t Au, Granodiorites have a CoG of 0.622 g/t Au, Mafics have a CoG of 0.665 g/t Au, Saprolite and Saprock have a CoG of 0.552 g/t Au.
- Associated metallurgical recoveries have been estimated as 93.8% for Felsic Tuffs, 95.2% for Granodiorites, 91.3% for Mafics and 96.4% Saprolite/Saprock
- Full mining recovery assumed.
- Reserves have no additional dilution added to that that inherent in the SMU of 5 m x 5 m x 5 m diluted mine block model.
- Reserves are based on a US$1,200/oz Au gold price.
- Reserves are converted from resources through the process of pit optimization, pit design, production schedule and supported by a positive cash flow model.
- The ore reserves were estimated by Bret C Swanson, BE (Min) MMSAQP #04418QP, a Qualified Person.
- The reserves are valid as of September 1, 2016.

16 Mining Methods

16.1 Parameters Relevant to Mine or Pit Designs and Plans

16.1.1 Geotechnical

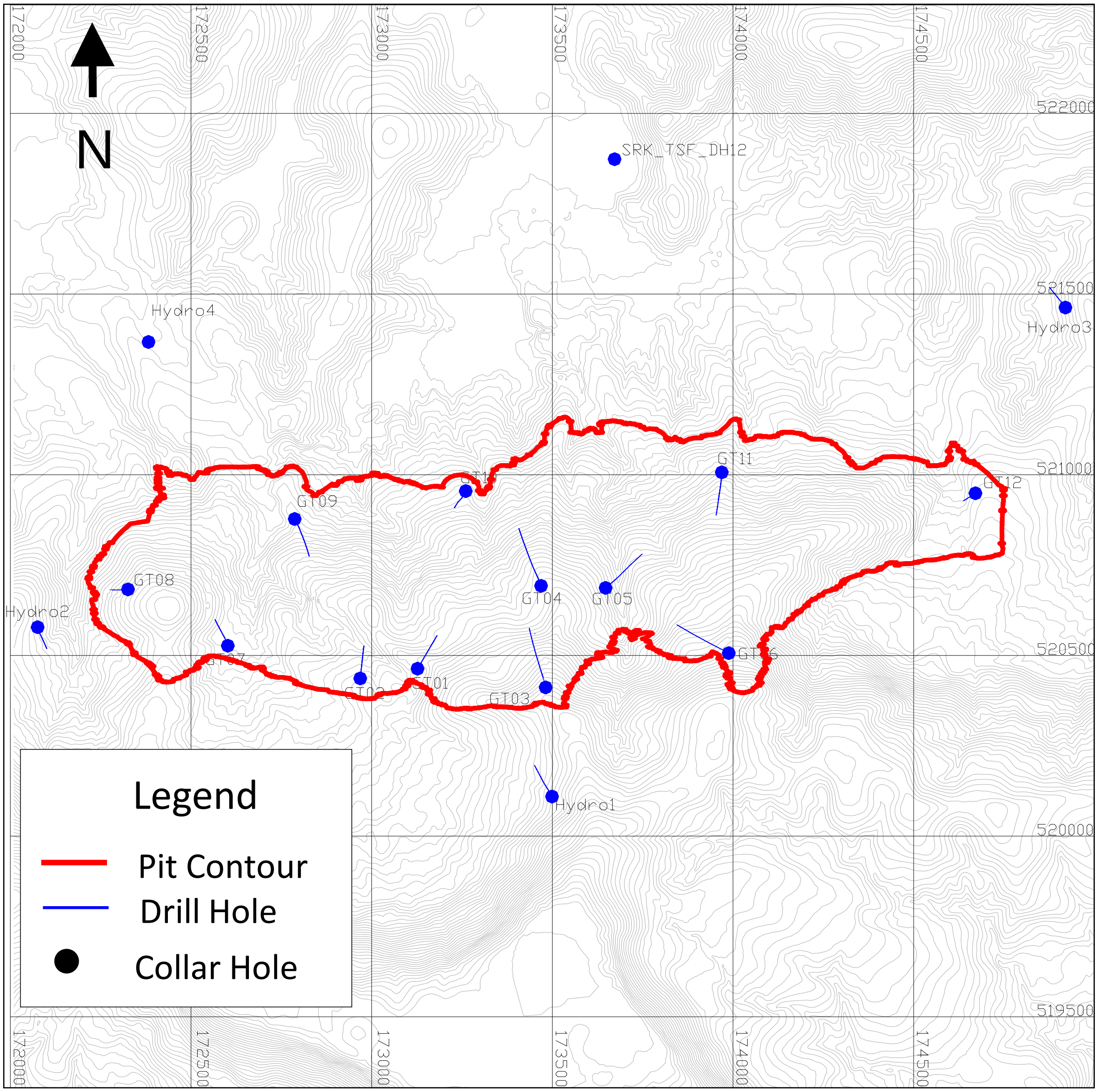
This section pertains to the geotechnical parameters and overall, interramp and bench scale stability analysis used in support of the reported reserves.

The open pit geotechnical field investigation consisted of seventeen drillholes, designed to examine rock mass fabric and structural features in and around the mineralized zone. Drillholes were drilled at varying orientations into the hangingwall, footwall, and mineralized rock to characterize the range of rock fabric variations. The field investigation included drilling of oriented core, geotechnical core logging, and core sample collection for laboratory strength testing. The complete field characterization program and results of analyses are available in the Montagne d'Or BFS Appendix 5.4 (Lycopodium, 2017).

Two major geotechnical domains have been identified in the Project. The first is a hard rock slope composed of strong foliated metamorphic rock and the second is a near surface saprolite soil domain that controls the stability of the upper 30 to 40 m of the ground. The saprolite is a deeply and intensely weathered residual rock that behaves like a soil. It is weak, nearly saturated, and easily deformable. Figure 16-1 illustrates the SRK BFS geotechnical drilling program which consisted of 3,800 m of core drilling.

The fresh hard rock consists of granodiorite, felsic tuff, mafic volcanics, and diabase dikes. Structural features (discontinuities) encountered during the field investigation consisted of joints, lithological contacts, veins, dikes, foliation, faults, shear zones, and fractures. The rock is characterized as strong to very strong with Unconfined Compressive Strength (UCS) values ranging from 80 to 200 MegaPascals (MPa). The rock is moderately jointed and has a very strong foliation joint set dipping south at approximately 70° throughout the deposit. RQD values are in the 90's and the rock mass rating ranges from 50 to 70, which indicates a fair to good quality rock mass.

A structural model was developed for the Project by Keith Benn. A total of 23 major structures were modeled using LeapFrog® software (ARANZ Geo Limited, 2014). These include two primary fault orientations that are near vertical and two sets of shear zones. The shear zones are geologic shear zones and consist of fresh strong hard rock as described above.



Source: SRK, 2017

Figure 16-1: Location of Geomechanical Drillholes

16.1.2 Slope Geometry

The BFS level slope geometry was based on bench, Interramp, and overall analysis. The slope design parameters are summarized in Table 16-1: .

Table 16-1: Open Pit BFS Level Design Parameters

Parameter	Unit	Single Bench Design in Fresh Rock (15 m)		Double Bench Design in Fresh Rock (30 m)	
		Footwall (North Wall) Value	Hangingwall (South Wall) Value	Footwall (North Wall) Value	Hangingwall (South Wall) Value
Maximum Overall Slope Angle (OSA)	°	45	41	49	49
Interramp Slope Angle (IRA)	°	49	45	54	49
Batter Face Angle (BFA)	°	70	64	70	64
Bench Height	m	15	15	30	30
Berm Width	m	7.5	7.5	10.5	10.5

Source: SRK, 2017

The maximum interramp slope height (bench stack height) is 150 m on the north wall. A ramp or geotechnical berm with a width of 14 m is required between bench stacks. The berm may be omitted on the south wall of the proposed pit. A minimum step-out of 5 m is required at the base of the saprolite/saprock for drainage and cleanup. No ramps are planned on the south highwall, however, a geotechnical bench will be necessary to catch multi-bench rockfalls.

Saprolite slopes will be critical for stability and should be excavated at a maximum 30° interramp angle. Bench height should be 5 m maximum. Benches should be graded at 2% to 3% laterally for the final design to promote drainage of stormwater off the benches. Vegetative cover should be established on all saprolite slopes to prevent saturation, failure by creep, erosion and gullying mechanisms.

The bench geometry is the limiting factor on interramp and overall angles for the pit. Geotechnical mapping should be undertaken to determine if the wall angles on the hanging wall (south wall of the pit) can be optimized and steepened once the initial benches in the hard-fresh rock are exposed.

The recommended slope geometry meets industry accepted slope acceptance criteria.

16.1.3 Analysis Method

The FoS for the reserve pit was computed using limit equilibrium methods using the method of slices. Slide 6.0 (version 6.029), a two-dimensional limit-equilibrium slope stability program (Rocscience, 2010), was used to evaluate the stability of the pit design. Spencer's method of slices was used to calculate the FoS as it computes the factor solving for both force and moment equilibrium.

Seven cross sections were analyzed by SRK assuming plain strain conditions. These methods assume that the shear strengths of the materials along the critical surface are governed by modified Hoek-Brown strength envelopes for the rock mass and linear Mohr-Coulomb strength envelopes for saprolite.

Both circular and non-circular searches were performed based on a path search algorithm. The path was defined to limit the search from the toe of the slope to the crest. In soil like materials such as the saprolite, the slope search in some cases identified critical surfaces internal to these units.

SRK used Sblock V2.022, developed by Esterhuizen, to analyze stability of the benches. Analysis results indicate well over 90% reliability for catching structurally controlled bench break failures with the proposed bench dimensions.

16.1.4 Input Parameters

SRK completed 7 major slope stability analysis runs to determine the FoS. A representation of the pit dimension, geology, hydrogeological model and structural information from 3-D modelling of the deposit were combined with the rock mass strength inputs required to simulate the pit wall FoS against failure.

The generalized Hoek-Brown strength criterion was used to represent the rock mass behavior and is expressed in terms of major and minor principal stresses (σ_1 & σ_3), and the material constant ($m_{i)}$ and are detailed in Table 16-2. These material constants have been derived from the laboratory strength test results.

Table 16-2: Rock Mass Strength Parameters

Model Material	Unit Weight (kN/m^3)	Uniaxial Compressive Strength (UCS (Mpa)	mi	Rock Mass Rating (RMR76)	a	mb	s	Cohesion (Mpa)	Angle of Internal Friction Φ (°)
Pelitic Sediment	29.8	83.5	20	62	0.502	1.325	0.0018	1.705	39
Mafic Volcanics	29.8	83.5	20	62	0.502	1.325	0.0018	1.705	39
Felsic Tuff	27.5	75.9	13	70	0.502	1.525	0.0067	1.961	39
Granodiorite	27.1	84.4	26	70	0.502	3.05	0.0067	2.348	46
Diabase	30.2	80.0	15	66	0.502	1.322	0.0035	1.776	39

Source: SRK, 2017

Table 16-3 is a summary of the saprolite strength inputs used in the stability analysis. These strengths were based on the triaxial laboratory testing results. For open pit slope stability analysis, the drained Mohr-Coulomb strengths may be used to assess slope stability analyses since the pit design includes bench drainage structures located at the base of the saprock elevation. Back analyzed material properties from natural slope failures yield similar results to laboratory and field index testing results.

Table 16-3: Rock Mass Strength Parameters Summary of Saprolite Strengths used in Stability Analysis

Strength	Cohesion (kPa)	Friction (°)	Undrained Shear Strength Cu0 (kPa)
25th Percentile	8	30	
Average (50th percentile)	23	30	
Undrained			60

Source: SRK, 2017

16.1.5 Overall Slope Analysis

The results of the overall slope stability analysis indicate the factors of safety exceed 1.8 for the overall slope geometry. The critical overall stability section is on the south wall of the pit and has a slope height of 308 m from the top of the open pit slope to the bottom of the pit. The piezometric level is just below the open pit surface. A tension crack has been assumed to exist in the analysis. The critical failure surface has a predicted minimum FoS of 1.80 and daylights at the toe of the pit slope. The critical surface is predicted to run predominantly through the felsic tuff and diabase dike units.

The north wall was also analysed for stability using limit-equilibrium analysis. The primary foliation is oriented in the dip slope on the north wall. The slope angles are shallower than the south wall and the overall slope heights are lower. Even with the foliation in the dip slope orientation on the north wall the FoS are higher than those presented for the sections on the south wall. The minimum FoS on the north wall are sufficiently higher than 2.0 on the overall slope for all sections analysed.

Stability of the overall pit slopes meets industry accepted slope acceptance criteria.

16.1.6 Saprolite Slope Stability

The saprolite slopes have been analyzed separately for stability to determine the minimum FoS on all cross sections. The saprolite slopes are the upper portion of the overall pit slope and each analysis section has been analysed independently of the global stability analysis. Strengths for the saprolite were developed from laboratory testing and back analysis of natural slope failures. The analyses assume that groundwater near the saprolite slopes are drained. If undrained conditions exist the slopes will fail by mechanisms that include erosion, flow, and creep.

The critical section is where the saprock contact dips towards the pit at approximately 15° to 20°. This section is located where the highest cut slope in the saprolite is planned, (~ 60 m high). The most likely critical failure surface runs from the tension crack at the crest of the slope and daylights at the base of the saprolite. The minimum FoS is 1.2. The FoS would be 1.4 based on average strengths. Sensitivity analysis was run on this section assuming pore water pressures acting with a Ru factor of 0.3. Under this condition, the FoS would drop to 1.0 approaching the limit of equilibrium. If the saprolite material is saturated there is the potential for movement.

The results of the slope stability analysis indicate that under anticipated conditions the slopes will remain stable at the recommended interramp angle of 30°. Even though the saprolite slope cuts have been designed to meet the slope acceptance criteria at a FoS of 1.3, some slope failure mechanisms might occur that are not addressed by stability analysis. When the saprolite cuts are exposed (i.e., un-revegetated) they will likely be subjected to creep deformation, erosion, and slump failure mechanisms. Failure mechanisms that include gullying, piping, and erosion will likely be exacerbated by precipitation onto exposed slopes that have not been sufficiently revegetated. Vegetative cover should be established on all cut slopes as soon after excavation as possible prior to the main rainy season. Berm surfaces should be laterally graded at 2° to 3° to assist drainage off benches.

Monitoring is recommended due to the uncertainty in slope conditions in the saprolite. A slope monitoring program should be implemented before mining and earthworks on the Project site. The slope monitoring program will be used to identify any incipient movement indicating the onset of failures and determine the appropriate course of action, which might include unloading or buttressing of slopes if a slide or failure is identified.

16.1.7 Waste Rock Dump Stability

The CWRD is located close to the pit limit and was subsequently identified as an area that required stability analysis to assess whether a geotechnical risk was evident. After investigation, SRK determined the minimum FoS to be1.40. The potential critical failure surface would be on the 20 m high berm at the base of the WRD (at 36°). This critical surface passes through the saprolite foundation and extends to the crest of the dump slope. The overall slope FoS is 1.70 for the 100 m high dump slope with an overall slope angle of approximately 24°.

16.1.8 Geotechnical Risk Mitigation Measures

The following risks to the Project have been identified and incorporated on the Project risk register. Mitigative measures have been incorporated into the pit design.

- The existing saprolite slopes and existing landslide hazard, as documented in the Rostan Report (Rostan, 2015), remain one of the highest risk to the Project. Engineering parameters collected and determined during this study indicate that stability may be achieved if appropriate drainage measures and vegetative cover are placed. However, geologic observations indicate saprolite failures and localized debris flows may occur in the natural terrain, even without mining activity. The recommended slope monitoring program will provide warning of saprolite movement or debris flows.
- Groundwater levels and flow at the base of the saprolite were observed in exploration drill pad sites. The stability of the saprolite will be a function of maintaining drained and depressurized conditions in the saprolite slopes.
- The groundwater levels in rock slopes is relatively high, however the overall slope stability is acceptable. Localized instabilities of plane shears or wedges may occur. The slope monitoring program should be able to identify potentially unstable areas. Appropriate remediation may be taken, including optimization to the mine design to local conditions.
- There is the potential for rockfall from bench faces. This hazard and risk is addressed by bench design and maintaining adequate catch benches (as per the BFS open pit design) as the mine is excavated.
- The structural model indicates that most of the structures are near vertical and therefore unlikely to form multi-bench wedge failures. The structural model is based on surface topography, geophysics, and core log data. The structural model and the potential for larger multi-bench failures from structural wedges should be confirmed by geologic mapping as the initial fresh rock benches are exposed during mining.

The bulk of the geotechnical data collection and analysis has been based solely on core data at the BFS level. There may be the potential for other failure mechanisms that have not been identified to date. Overall geotechnical risks to the Project can be reduced by implementing the recommended slope monitoring system and conducting geotechnical mapping and analysis as mining commences in the Montagne d'Or open pit.

16.1.9 Hydrologic and Hydrogeologic

Significant volumes of surface run-off and shallow groundwater from the drainages where saprock is exposed will be captured in a diversion ditch along the top of the pit, to minimize the volume of water reaching the exposed rock in the open pit. The diversion water will be routed to sediment control ponds and undisturbed creeks. However, groundwater in bedrock and in faults and joints within the bedrock will report to low points in the open pit and require pumping to a CWP. Because the intact bedrock is of low hydraulic conductivity, the relative contribution of groundwater reaching the open pit will be less than that of surface water run-off reporting to the pit.

The model predicts maximum passive groundwater inflow up to 3,975 m^3/d (46 L/sec) during Year 10 of pit excavation. An average predicted pit inflow through LoM is 2,250 m^3/d (26 L/sec). Total maximum annual pit inflow, considering both net precipitation/surface run-off and groundwater flow, is predicted at 8,800 m^3/d (102 L/sec). The average annual total inflow to the pit following closure is predicted to be 5,668 m^3/d (65.6 L/sec). Approximately 40% of the predicted total inflow is coming from groundwater and the remaining water is sourced from direct precipitation and run-off. Predicted sources of groundwater inflow to the pit are 1) groundwater in saprock that primarily discharges to the pit from the southern highwall and 2) depletion of groundwater storage.

Because the long and intense wet season of the region, surface water inflows to the pit, both from run-off from the exposed pit walls and run-on from upgradient areas that cannot be feasibly diverted around the pit, will report to the pit bottoms along with groundwater inflows and will accumulate until it can be evacuated by pumping to the pit rim and then to the contact water management system. The mine water management plan includes a pumping system designed to evacuate the pit bottoms as rapidly as possible, but accumulation in pit sumps during the wet season is unavoidable. Mining activities should incorporate contingency plans to address the possibility of pit flooding and heavy run-off on the pit walls during periods of intense rainfall.

When mining ceases, the open pit will fill with a combination of groundwater and a predominant amount of run-off and direct precipitation. The initial groundwater contribution will be about 40% of the total inflow. Groundwater inflow will decrease as the lake fills, and will comprise a small component of inflow once the pit lake reaches the overflow point. Once the pit lake reaches the overflow point of the pit, it will be routed to undisturbed drainages, as the pit lake water quality is expected to be suitable for discharge.

16.2 Mine Design

The Montagne d'Or mine will be an open pit mine that uses gravity/cyanidation as the primary method of extracting gold from the Mineral Resource. Through the process of pit optimization, pit design, production scheduling, and capital and operating cost estimation, the conversion of Mineral Resources to Mineral Reserves resulted in a diluted reserve of 2.75 Moz Au at 1.58 g/t Au defined in situ before metallurgical recoveries.

The open pit is approximately 2.5 km long by 500 m wide, and of varying depth from surface, with a total volume of 112.5 Mm^3 and a stripping ratio of 4.5 to 1 (waste to ore). (Note 1: The open pit is located on the side of a hill. The average pit north wall is approximately 125 m deep from original ground surface, and the average pit south wall is approximately 225 m in height. The pit centroid depth from original ground surface is 185 m). Figure 16-2 illustrates the pit design, dump design and expected tailings location for the Project.

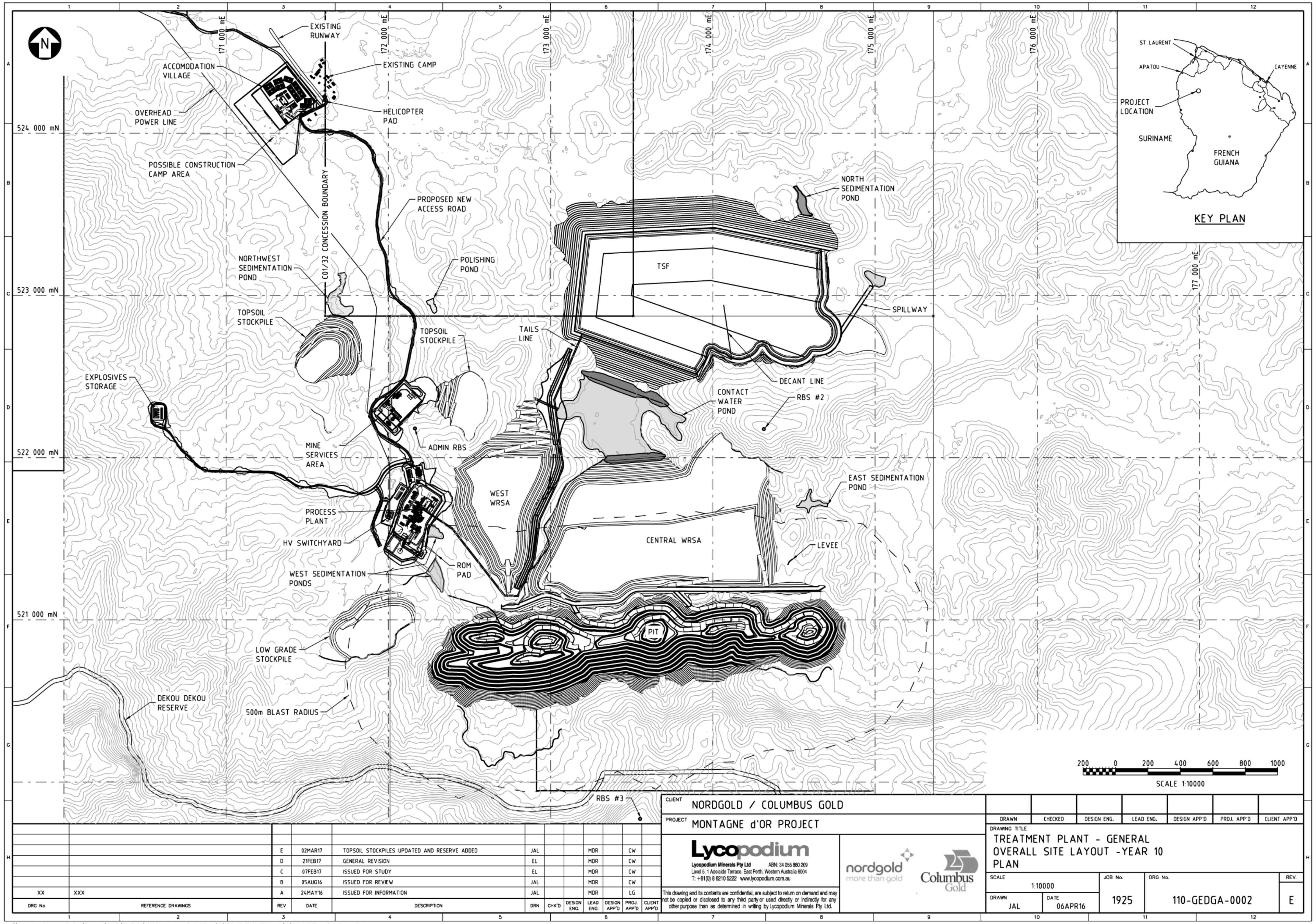


Figure 16-2: Montagne d'Or Site Layout

The mine production schedule is based on feeding the processing facility operating at a rate of 12,500 t/d or approximately 4.6 Mt/y of mill feed. The mill feed was broken into three CoGs that represent gold prices of US$400/oz, US$800/oz and US$1,200/oz, and includes multiple recoveries ranging from 90.3% to 96.4% dependent on rock types, for the purpose of the CoG calculations.

The planned mining rate targets approximately 80 kt/d (waste and ore), which provides a higher mill feed can process, requiring mill feed stockpiles to be used to store the excess. The use of stockpiles ensures that the highest grade mill feed is sent to the crusher before lower grade is processed. This creates a variable cut-off that defers marginal mill feed that will be processed at the end of the mine life, thus optimizing the Project NPV and cash flow. The maximum stockpile size is approximately 8 Mt of material. Mining rates have been adjusted by up to 30% to account for the wet and dry seasons that will be encountered during operations.

Dilution has been incorporated into the mine block model for the BFS. As there is no operational history, dilution was calculated by determining the partial quantity of gold units within and outside the grade shell used for resource interpolation. The diluted grade for the model is referenced to a 5 m x 5 m x 5 m block dimension that represents the SMU assumed for the BFS. This is supported by the planned drilling pattern of 5.1 x 5.1 m representing grade control definition.

16.2.1 Pre-Production

The pre-production period has been modelled by SRK on a monthly basis for the earthworks and pit progression. After several iterations of the pre-production mine schedule that looked at pre-production daily target rates of 5 kt/d, 10 kt/d, 15 kt/d and 25 kt/d, the 10 kt/d schedule was selected for the following reasons:

- The capitalized stripping was reduced to a minimum that still exposed hard ore ready for 91 t truck capacity implementation in Year 1;
- Enough hard rock was available for site construction of the plant, tailings and water structures;
- The mining rate, while limited to approximately two benches per month, will enable proper pioneering of mine access and infrastructure;
- Adequate training period for staff;
- Ability to commence low strip ratio, higher grade ore during payback period; and
- Establishment of low grade stockpile in orderly fashion.

Much of the open pit, WRD and site infrastructure footprint areas are heavily wooded with tall trees, scrub, wetlands and disturbed ground. The quality of sellable hardwood is unknown at this time, and any cash value of hardwood was not considered in the model.

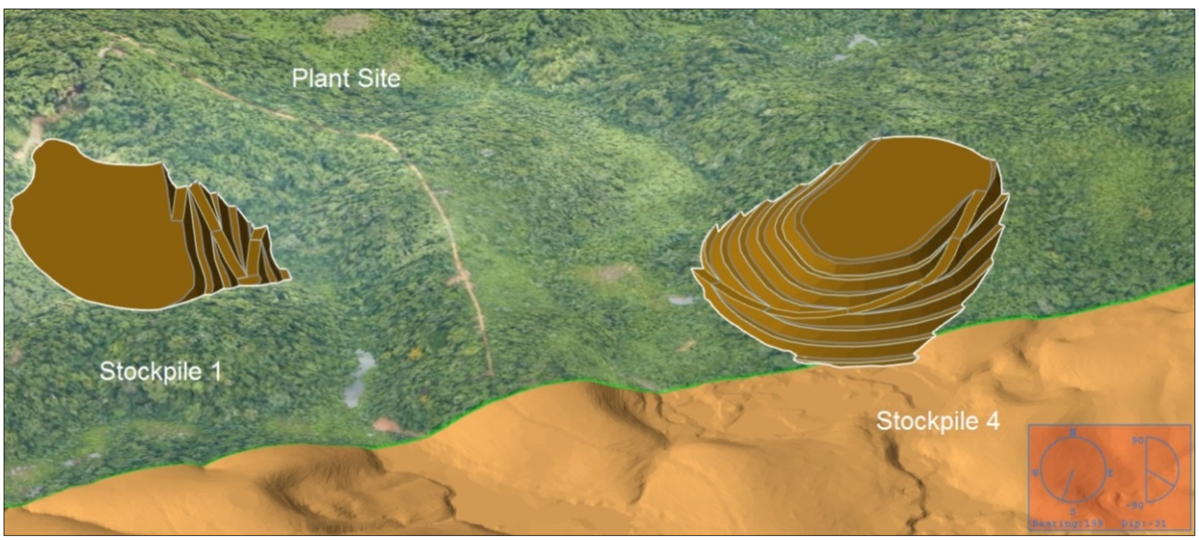
SRK has designed two soil stockpile locations that are located within closed catchments that will receive little additional surface water other than precipitation. The stockpiles are sized to contain the volume equivalent to a 50 cm thick topsoil layer over the entire disturbed area. Table 16-4 details the topsoil disturbance estimate.

Table 16-4 Topsoil Disturbance Estimate

Site Area	Volume (50 cm) m^3	Area m^2	Tonnes	Ha
Access	34,749	69,498	52,124	6.95
Frontage Road	22,905	45,811	34,358	4.58
LG Stockpile	91,765	183,531	137,648	18.35
Pit Limit	598,403	1,196,806	897,605	119.68
Process Plant	214,077	428,154	321,116	42.82
Stockpile 4	57,636	115,271	86,453	11.53
Stockpile 1	45,259	90,518	67,889	9.05
Tailings	853,339	1,706,679	1,280,009	170.67
Tailings Access	48,972	97,943	73,458	9.79
CWRD	572,605	1,145,209	858,907	114.52
WWRD	214,574	429,148	321,861	42.91
Total	**2,754,284**	**5,508,568**	**4,131,426**	**550.86**
Approximation of Disturbance Site	**3,827,572**	**7,655,145**	**5,741,359**	**765.51**

Source: SRK, 2017

Figure 16-3 shows the location of the ultimate soil stockpiles only. The stockpiles will likely be dynamic in nature as it is likely that during operations topsoil will be used for progressive rehabilitation during operations at the same time as additional soil is added due to further disturbance around the site. It is unlikely that large areas will be cleared for a considerable time before operations as the vegetation is anticipated to play a pivotal role in stabilizing saprolite in the region.



Source: SRK, 2016

Figure 16-3: Perspective View of Soil Stockpile Locations

16.2.2 Pit Optimization

SRK used the Whittle™ pit optimization software to determine the optimal size of the BFS reserve. It should be noted that between the PEA and the BFS, in-fill drilling was carried out primarily for the conversion of Indicated to Measured resources, and no deep drilling that may materially change the pit was conducted. The two main concerns that were considered in conjunction to the economic results produced by Whittle™ were the geotechnical stability of the saprolite and saprock slopes, and the proximity of the pit crest to the Dékou-Dékou Massif Reserve.

Most of the trade-off studies concerning pit size and production rate were evaluated in the 2015 PEA for the Project (SRK, 2015). As such, the pit optimization for the BFS focused on minimizing reserve risk given the high level of incremental stripping inherent in the deposit, and understanding the effects of geotechnical risk on the pit shell used as the basis for pit design.

Table 16-5 illustrates the SRK parameters used for the pit optimization for the deposit. The capital and operating costs were estimated at the time of the pit optimization and therefore will not necessarily match those in the final report.

Table 16-5: Pit Optimization Inputs

Whittle™ Parameter	Type	Common Parameters	BFS Case
Mining Cost	Reference Mining Cost (US$/t)		$2.25
Geotechnical Slopes	Bearing/Slope	SAP/SAPR	30°
		Other	0/50°, 180/49°
Processing Cost			
	Process Name	Mill	
	Selection Method	Cut-off	
	Process Cost (US$/mill-t)	SAP	$10.76+$4
		SAPR	$10.76+$4
		FLTF	$12.2+$4
		LPTF	$12.2+$4
		GRAN	$12.71+$4
		FLPY	$12.71+$4
		QTZP	$12.71+$4
		METS	$13.23+$4
		AMPS	$13.23+$4
		MFVL	$13.23+$4
		WAST	$13.23+$4
	Process Recovery (%)	SAP	96.4%
		SAPR	96.4%
		FLTF	93.8%
		LPTF	93.8%
		GRAN	95.2%
		FLPY	95.2%
		QTZP	95.2%
		METS	91.3%
		AMPS	91.3%
		MFVL	91.3%
		WAST	91.3%
Ore Selection Method		Cut-off	Internal
Revenue and Selling Cost	Au Price(US$/oz)		US$ 1,200
Royalty, Refining, Transport etc.	Au Selling Cost (US$/oz)		$55.08 + $8
Optimization	Revenue factor range		0.0-1.5 76 factors
Operational Scenario – Time Costs			
	Initial Capital Cost		US$300 million
	Discount Rate Per Period		8%
Operational Scenario – Limits	Process Limit (t/y)		4,500,000

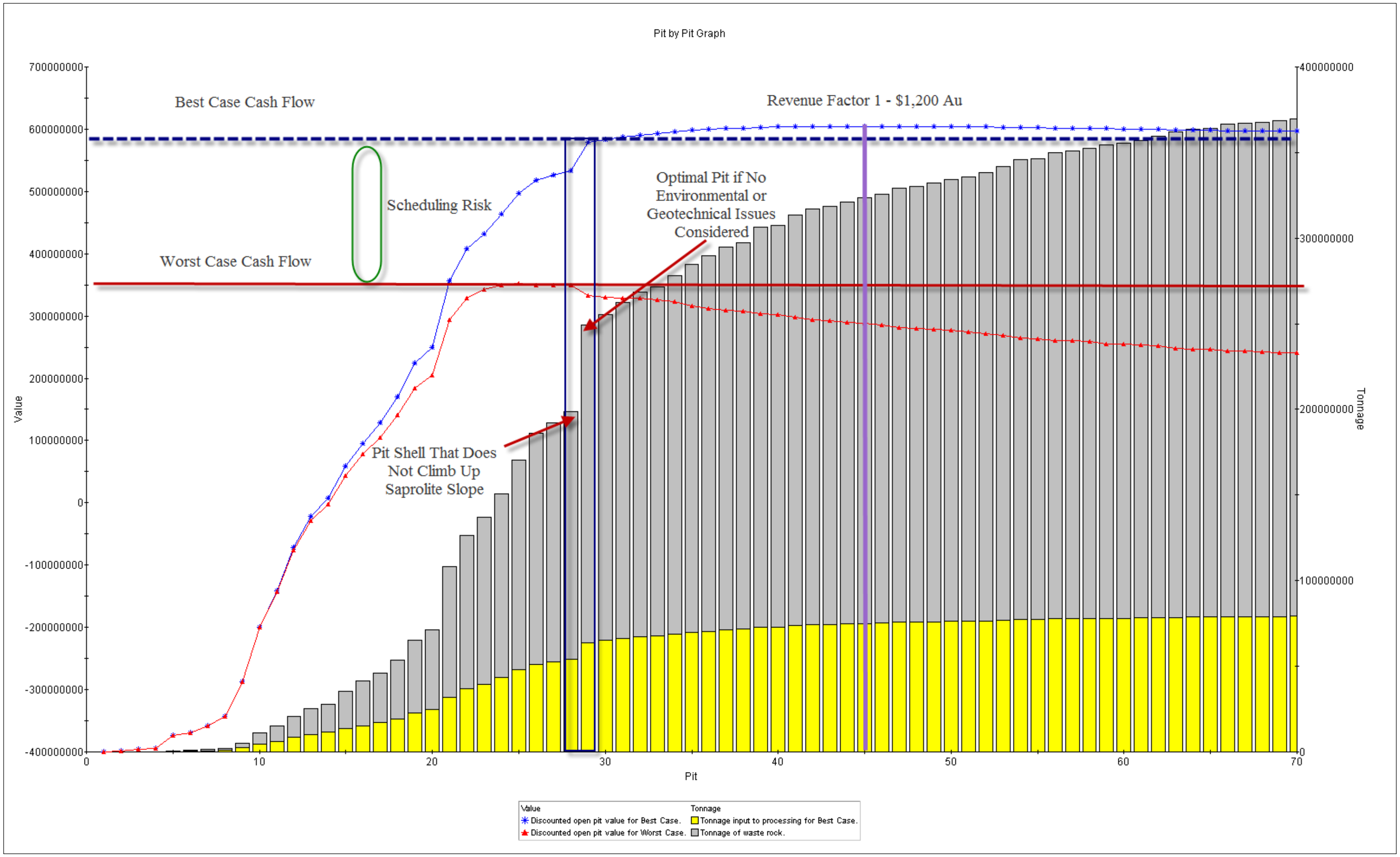
Source: SRK, 2016

In determining the optimum pit design, a combination of factors were considered to balance geotechnical concerns of excessive saprolite exposure on the final pit walls, provide a relatively conservative pit that minimizes incremental stripping, and scheduling risk while maximizing the reserve potential.

Figure 16-4 illustrates the relationship of ore to waste at varying prices for gold and the guidance cash-flow based on US$1,200/oz Au. Of particular note, is the reasonably large jump in waste for a modest increase in ore with minimal increase in best case cash flow between Pits 28 and 29. In addition, the worst case cash flow line does not drop until Pit 28, but does so at Pit 28. Because there is little incremental gain from a best case scenario and mining risk increases beyond Pit 28, SRK selected Pit 28 as the pit shell to base the reserve pit design. The added advantage of selecting this pit is it does not lead to additional saprolite exposure as the potential pit crests start paralleling topography as the gold price increases.

SRK did not include modifications for incremental bench mining cost above or below a reference bench. (Full haulage costs have been included in the fleet estimation and mining cost estimate detailed in the mining operating costs.)

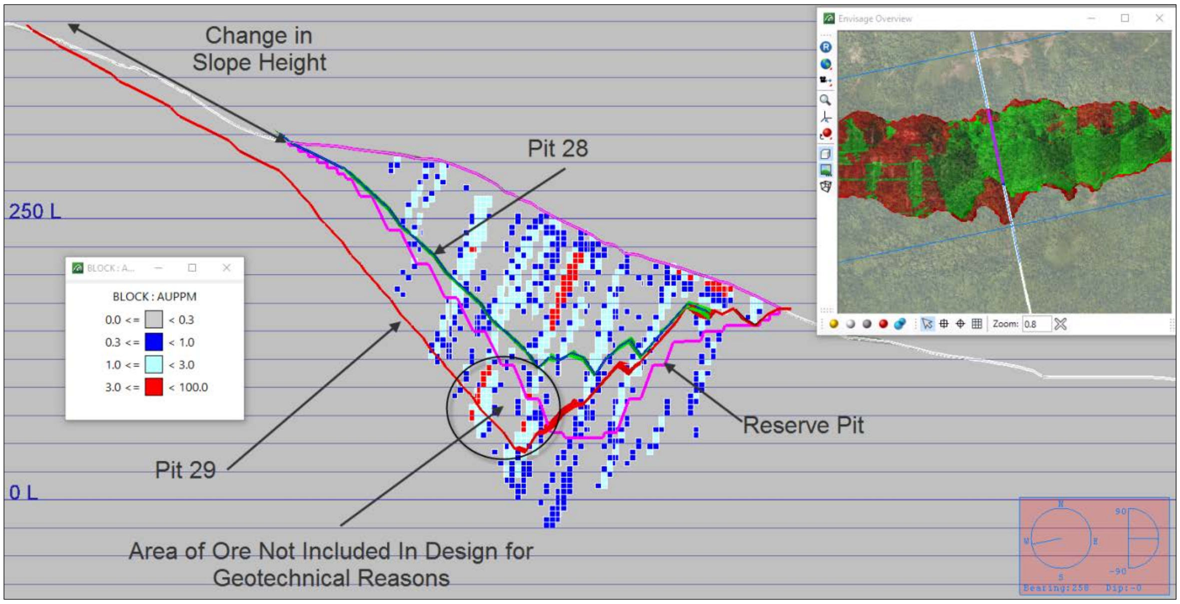
The economic assumptions used for pit optimization may not necessarily match those used in the final BFS economic model.



Source: SRK, 2016

Figure 16-4: Pit by Pit Analysis

Figure 16-5 shows the spatial effect of moving from Pit 28 to Pit 29 when compared to the final SRK pit design. The SRK reserve matches the Pit 28 crest, but misses the high grade (yet deep) ore and extends outside both pits due to mining width and pit access on the low wall side of the pit. From the overview of the section, the middle portion of the pit shows the greatest variability as Pit 28 and Pit 29 are reasonably consistent in other parts of the deposit, and particularly in the western portion.



Source SRK 2016

Figure 16-5: Whittle™ Pit Selection

16.2.3 Pit Design

The Montagne d'Or pit design is defined by a relatively high pit wall on the south side of the deposit that intersects hard and soft saprolite rock. The south wall is complicated by the interface of the 30 m benches with saprolite that is uneven and varies in depth up to the pit crest. The south wall toe location is a primary driver to the economics of the pit as the stripping ratio is highly sensitive to changes with the pit toe location.

Ramps are placed in the north wall of the pit where changes to the stripping ratio is relatively minor. Due to the length of the pit, there are four ramps that do not overlap each and provides access to the WRD, low-grade (LG) stockpile and RoM pad. Due to the undulations and different elevations of the pit exits, SRK designed a frontage road that ties into the pit exits and allows for a 1% drainage from the eastern pit rim to the west, protecting the watersheds to the north of the pit. This also allows for increased haul speeds of trucks delivering ore to the crusher.

A 23 m pit ramp width was set for 91 t class haul trucks and 14 m for 40 t ADTs, with the latter allowing single lane access roads for the larger trucks. The ramp to truck width is a 3.3 times factor assuming a 6.9 m operating width for the 91 t class trucks. Ramps are limited to a 10% grade based on the shortest ramp distance; this means that ramps should never exceed 10% in the pit design. Single lane roads of 14 m for a 91 t truck are generally limited to three benches at the pit bottom.

Pioneering work is limited to ADTs so the ramps are generally 14 m wide when saprolite is encountered in the pit.

The pit design parameters applied are detailed in Table 16-6.

Table 16-6: Pit Design Parameters

Parameter	Unit	Saprolite	Fresh High Wall	Fresh Foot Wall	Dumps and Stockpiles
Overall Slope Angle	°	30	49	54	23
Batter Angle	°	85	64	70	37
Berm Placement Height	m	5	30	30	10
Flitch (Mining Face) Height	m	5	5	5	10
Berm Width	m	8.22	10.5	10.5	10
Ramp Width – 2 way	m	23	23	23	23
Ramp Width – 1 way	m	14	14	14	NA
Ramp Gradient (Shortest)	%	10	10	10	10

Source: SRK, 2016

Figure 16-6 illustrates the BFS pit design used in the evaluation of the Project.

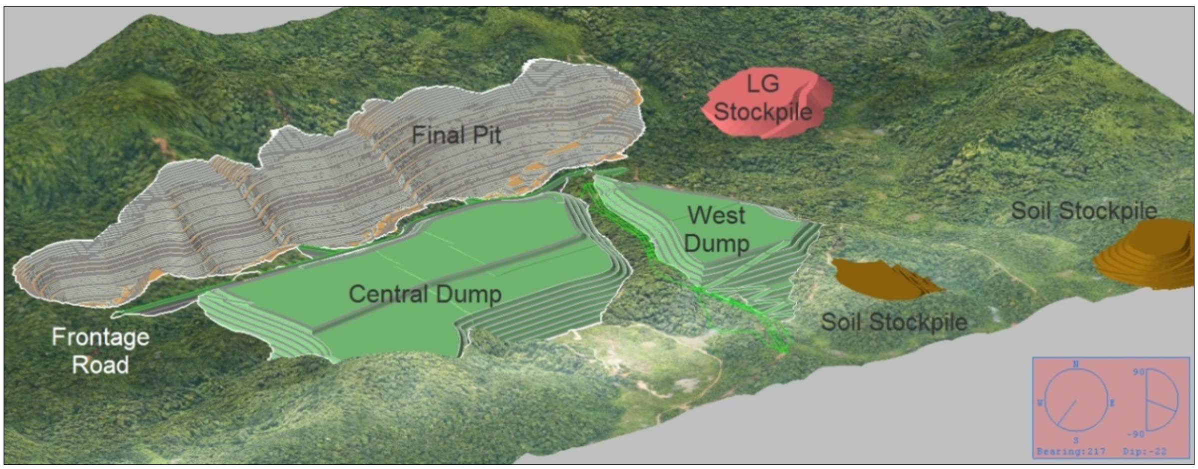
Source: SRK, 2016

Figure 16-6: BFS Pit

The SRK pit design is considered to be a feasibility-level design as the exploration program focused on geological and resource expansion. As a result, there has been no mine planning (targeted) drilling that can potentially straighten pit walls or remove multiple pit bottoms. SRK would recommend that during detailed engineering and further exploration, a fence of drillholes target the pit toe on the south wall given the sensitivity of the south wall to stripping ratio and final crest location arising from the problems with expanded saprolite exposure.

16.2.4 Dump Design

SRK has designed two WRDs and the tailings embankment bulk earthworks for placement allocation of waste. SRK designed two soil stockpiles for the top 50 cm of soil removed during mining operations. Figure 16-7 illustrates the dump layout in relation to the reserve pit.



Source: SRK 2016

Figure 16-7: General Waste Rock Dump Layout

The WWRD volumetrics have been calculated using an assumed loose density of 2,000 kg/m^3, or a specific gravity (SG) of 2.00. For hard rock, this equates to a 40% swell factor which is generally high, but because the WRD will likely be co-mingled with saprolite SRK believes this density to be prudent. It should be noted that there is an opportunity to extend the dumps between the WWRD and CWRD as there are no topographical limitations to expansion.

WRDs will be placed in 10 m lifts with a 10 m berm. The berms reduce the velocity of water running on the dump face, thus reducing problems relating to dump face erosion from the heavy rainfall events in the region, and to improve overall geotechnical stability of the dump. The ultimate bench and berm configuration gives an overall slope angle of 23^0 for efficient closure. Because there is no limitation to available dump space, this angle could be reduced further to facilitate closure without significant disruption to operations.

The general design criteria for the various dumps are defined as follow:

- Dumps start in the WWRD and in starter area of the CWRD;
- When the WWRD is finished, the CWRD expands to the east allowing progressive rehabilitation;
- Rock dumps have shallow overall slopes to minimize regrading costs for closure;

- Rock dumps are located at the top of water catchments to minimize water conveyance, treatment (if needed) and turbidity reduction; and
- Slopes heights generally under 80 m in height and thus have reduced post mining topography influence on surrounding terrain.

The general design criteria for the WWRD includes:

- Valley fill dump with dump toe buttressed by natural topography;
- Water reports to a confined catchment allowing for collection and treatment prior to release;
- Foundation is undisturbed saprolite for enhanced water percolation control; and
- Transportation of western pit waste to west dump is short and reasonable flat.

The general design criteria for the CWRD including Central Starter Waste Rock Dump include:

- Rock dumps placed on historically disturbed area;
- Limited to central water catchment with good sediment control in place; and
- Large generally low dump height aiding in ability to create NAG and PAG cells and paddock dumping of PAG.

The general design criteria for the Soil Stockpiles include:

- Located in self-contained catchments with minimal water running onto stockpile or stockpile toes;
- Centrally located to WRD and tailings limits; and
- Has the ability to expand and contract in volume for addition of soil and removal for rehabilitation purposes.

The general design criteria for the LG Stockpile include:

- Close to primary crusher;
- Located in self-contained water catchment with controls available downstream of the plant site;
- Ability to expand based with variation in CoG;
- May be relocated onto WWRD if mineralization is found to continue to the west of the current pit; and
- Shortest waste haulage distance for majority of pit.

16.2.5 PAG Management

SRK has estimated that approximately 41% of the waste material will be PAG material primarily coming from the felsic tuff and lapilli tuff rock types in the pit. As a result it will be important that operational procedures be put in place that minimize the time potentially acid generating rock is exposed to heat, oxygen and water. The most effective method of minimizing acid generation is to limit the oxygen available in the WRDs and this is primarily achieved through encapsulation of at least 5 m of NAG material within the lag time for PAG oxidation. The major encapsulation and PAG control methods include:

- Creation of PAG cells that can be covered with NAG within the lag time before oxidation;
- Paddock dumping of PAG rock at the toe of an advancing dump face. This is a co-mingle method;

- Compaction of saprolite cover layer to reduce water and oxygen ingress into the WRD; and
- Changing the dump lift height to 5 m thus increasing the compaction of the dump as trucks and bulldozers reshape the dumps.

Due to the topography of the site, sub-aqueous deposition will not be possible for the current pit footprint. If the orebody were to expand along strike then it may be possible to backfill old pits and create pit lakes to prevent oxygen reaching the reactive sulphide minerals in the PAG rock.

16.3 Mine Planning

16.3.1 Grade Tonnage

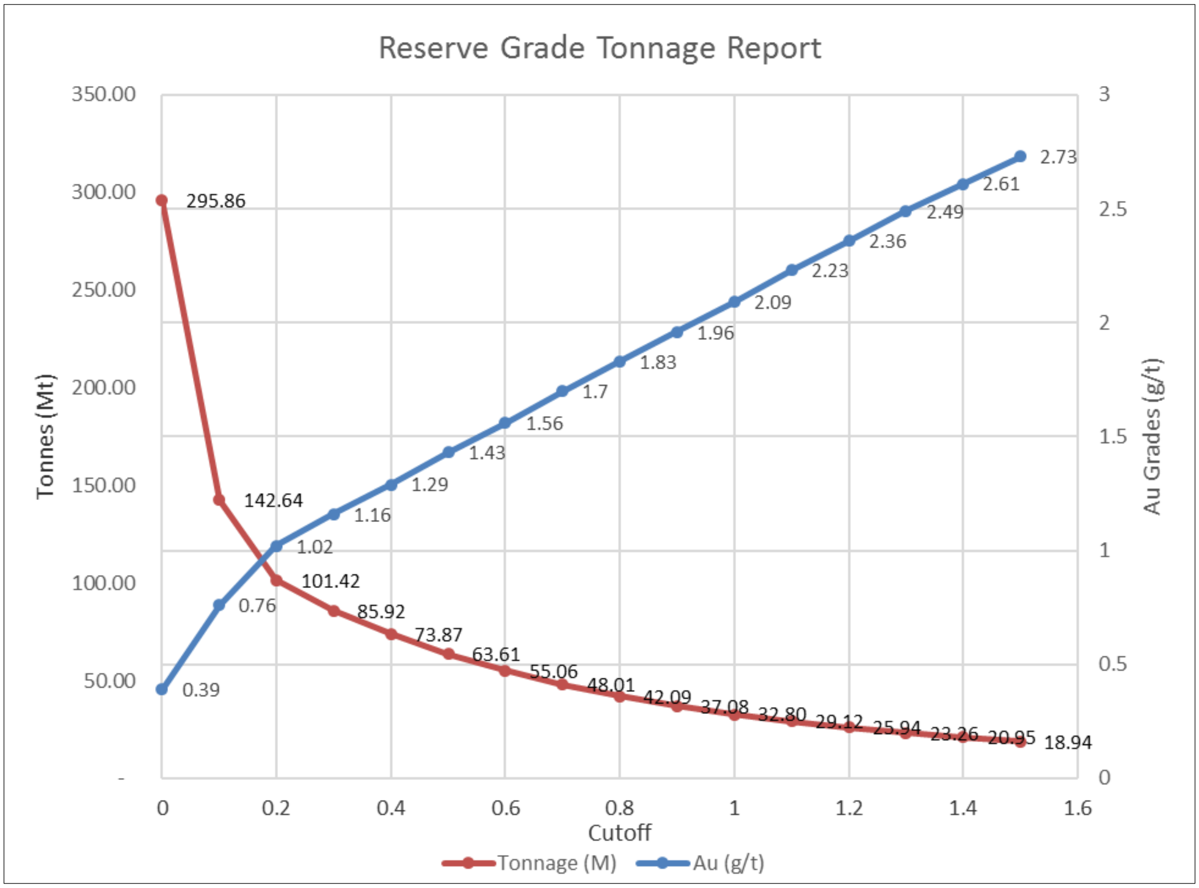
The grade tonnage has been limited to Measured and Indicated resources within the SRK reserve pit. The grade tonnage shows that as the grade increases from 1.5 g/t through 2.5 g/t Au, the stripping ratio increases from approximately 4:1 to 12:1. While the current reserve pit balances reserve size with mine life, there is the potential to modify the operating parameters to increase and decrease grade according to operational parameters encountered during operations.

The grade tonnage for the pit is detailed in Table 16-7 and is visually displayed in Figure 16-8.

Table 16-7: Grade Tonnage Curve within Pit Design

Cut-off	Au	Tonnage (Mt)	Ounces (Moz)	Stripping Ratio (W:O)
-	0.39	295.86	3.71	-
0.10	0.76	142.64	3.49	1.07
0.20	1.02	101.42	3.33	1.92
0.30	1.16	85.92	3.20	2.44
0.40	1.29	73.87	3.06	3.01
0.50	1.43	63.61	2.92	3.65
0.60	1.56	55.06	2.76	4.37
0.70	1.70	48.01	2.62	5.16
0.80	1.83	42.09	2.48	6.03
0.90	1.96	37.08	2.34	6.98
1.00	2.09	32.80	2.20	8.02
1.10	2.23	29.12	2.09	9.16
1.20	2.36	25.94	1.97	10.41
1.30	2.49	23.26	1.86	11.72
1.40	2.61	20.95	1.76	13.12
1.50	2.73	18.94	1.66	14.62

Source: SRK, 2016



Source: SRK, 2016

Figure 16-8: Grade Tonnage Curve within Reserve Pit

16.3.2 Cut-Off Grade

SRK selected three CoGs representing a high grade (US$1,200/oz), mid-grade (US$800/oz) and low grade (US$400/oz) gold price for calculating reserves in the Montagne d'Or mine plan. A breakeven CoG was used rather than the internal CoG and estimates were made before the final economic model was created.

Table 16-8 details the internal CoG for the high-grade (HG), mid-grade (MG) and low-grade (LG) categories.

Table 16-8: Cut-Off Calculations and Grade Bins

Description	Units	LG 1200 CoG	MG 800 CoG	HG 400 CoG
Common Assumptions				
Price				
Gold Price	US$/oz	$1,200	$800	$400
Gold Price	US$/g	$38.58	$25.72	$12.86
Smelting & Refining	US$/oz	$8.00	$8.00	$8.00
Royalty (NSR)	%	5%	5%	5%
Costs				
Smelting & Refining	US$/t milled	$0.1489	$0.2241	$0.4531
Royalty	US$/t milled	$1.0251	$1.0288	$1.0398
Mining	US$/t mined	$2.25	$2.25	$2.25
Other Costs (e.g. Reclamation)	US$/t milled	$0.2	$0.2	$0.2
G&A	US$/t milled	$5.78	$5.78	$5.78
Variables and Cut-Off				
Felsics				
Au Recovery	%	93.8%	93.8%	93.8%
Processing	US$/t milled	$12.93	$12.93	$12.93
CoG – Head Grade	g/t	0.617	0.929	1.878
CoG – Recovered Grade	g/t	0.579	0.871	1.761
Granodiorite				
Au Recovery	%	95.2%	95.2%	95.2%
Processing	US$/t milled	$13.4	$13.4	$13.4
CoG – Head Grade	g/t	0.622	0.936	1.891
CoG – Recovered Grade	g/t	0.592	0.891	1.801
Mafics				
Au Recovery	%	91.3%	91.3%	91.3%
Processing	US$/t milled	$13.97	$13.97	$13.97
CoG – Head Grade	g/t	0.665	1.001	2.023
CoG – Recovered Grade	g/t	0.607	0.914	1.848
Saprolite/Saprock				
Other Costs (e.g. Reclamation)	US$/t milled	$0.1	$0.1	$0.1
Au Recovery	%	96.4%	96.4%	96.4%
Processing	US$/t milled	$11.32	$11.32	$11.32
CoG – Head Grade	g/t	0.552	0.831	1.680
CoG – Recovered Grade	g/t	0.532	0.801	1.619

Source: SRK, 2016

16.3.3 Dilution

Resources were based on the SRK resource block model, constructed using a 10 m x 5 m x 5 m block dimension. To address the resource model and make it suitable for mine planning, SRK re-estimated the block model using a 5 m x 5 m x 5 m block dimension that corresponds to the Selective Mining Unit (SMU) of the operation. The block model was then estimated inside and outside the grade shell model and the partial gold from each estimation averaged. This created a fuzzy boundary of blocks along the edge of the grade shell that the CoGs are applied to. The end result is a diluted block model that better represents the grade control drilling pattern that can be selectively mined with the drilling pattern and loading equipment selected.

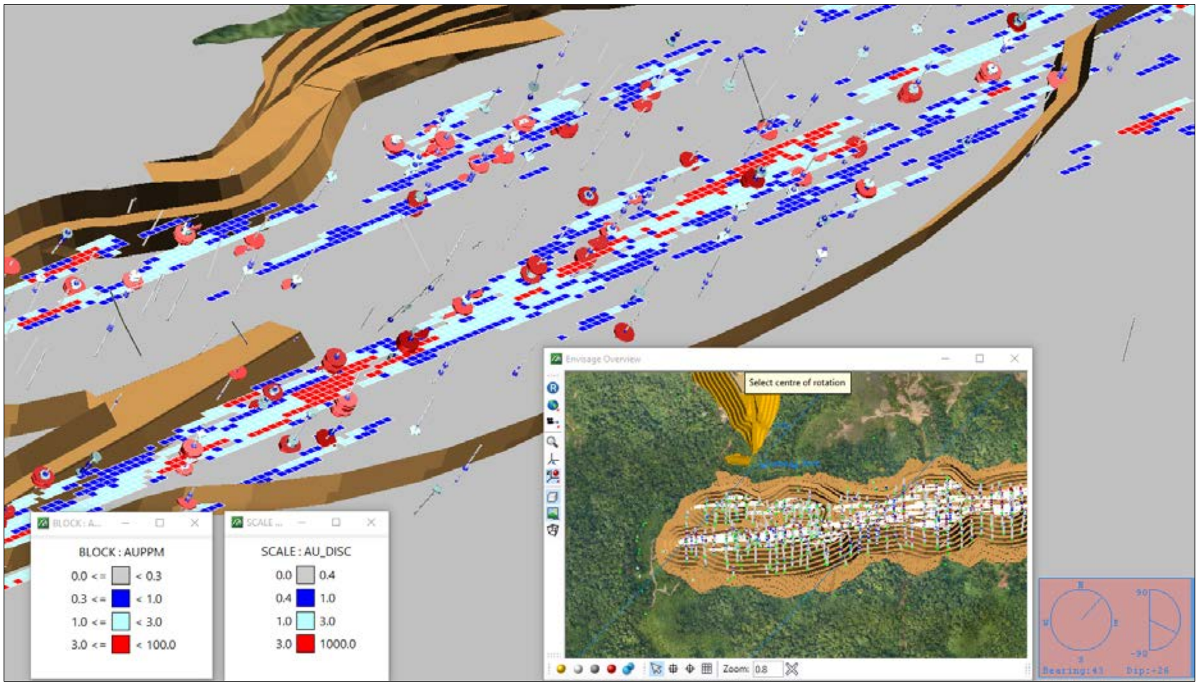
Table 16-9 summarizes the resource model compared to the diluted mine model. Only Measured and Indicated resources are compared.

Table 16-9: Comparison of SRK Resource Model to Diluted Mine Model within Reserve Pit

	Mine Model (5x5x5)	Mine Model (5x5x5)	Mine Model (5x5x5)	Resource Model (10x5x5)	Resource Model (10x5x5)	Resource Model (10x5x5)	Comparison	% Au Grade Dilution
	Ore Tonnes t	Au Ounces Oz	Au Grade g/t	Ore Tonnes t	Au Ounces Oz	Au Grade g/t	%Au Ounce Dilution	
Reserve Pit	54,113,441	2,745,390	1.58	55,138,157	2,976,174	1.68	8%	6%
High Grade (hg400)	13,379,875	1,364,730	3.17	15,195,435	1,593,169	3.26	14%	3%
Mid Grade (mg800)	22,590,658	940,100	1.29	23,493,250	982,036	1.30	4%	1%
Low Grade (lg1200)	18,142,908	440,560	0.76	16,449,472	400,969	0.76	-10%	0%

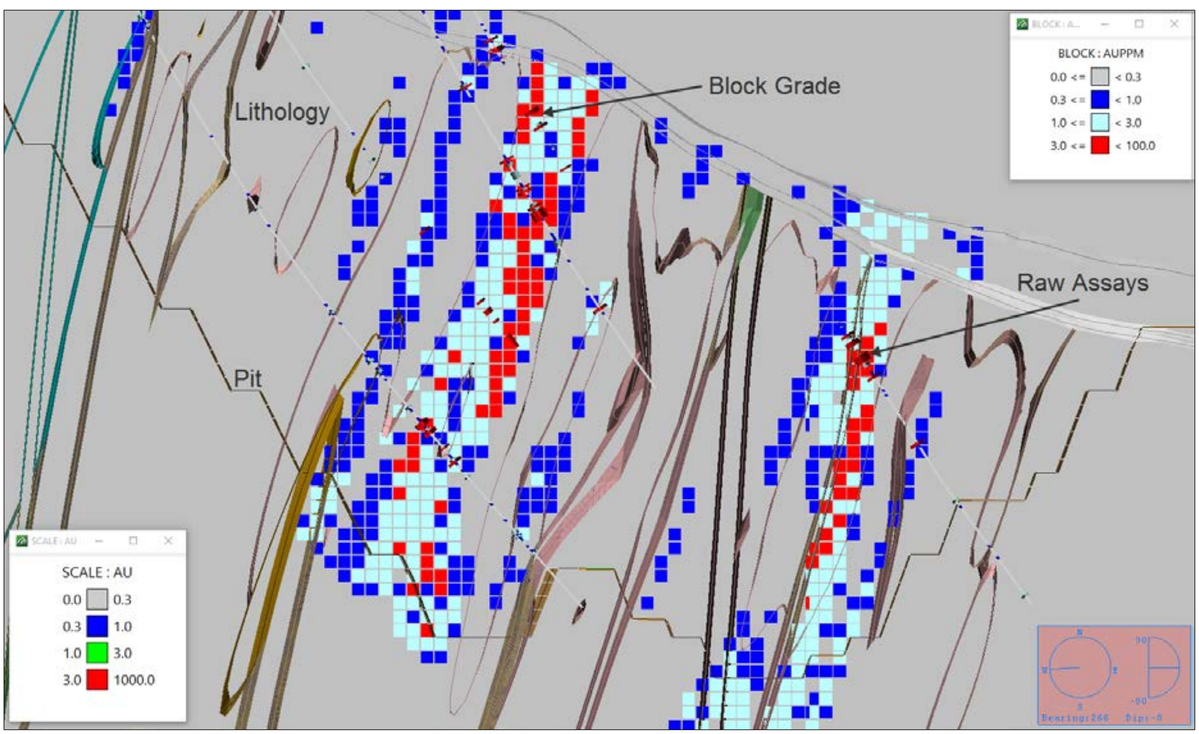
Source: SRK, 2016

From a contained ounce perspective, the HG and MG blocks were diluted on both a tonnage and a grade basis, but the LG portion of the blocks increased in tonnage while the grade stayed the same. The high grade zones are generally thinner than the low grade zones, so it is consistent to think the smaller SMU size reduces the high grade interpolation outside the grade shell. Similarly, the HG gold grade portion is diluted by 3%, but with the increase in low grade tonnage the weighted grade dilution is 6% on a reserve basis. SRK did not add any additional dilution or mining recovery modifiers to the reserve tonnes and grade. It should be noted that any Inferred blocks within the pit design shape were given zero grade and treated as waste. Figure 16-9 and Figure 16-10 show examples of the raw drillhole information, the variability in grade and the expression of the drillhole grade into the mine block model from a visual perspective.



Source SRK 2016

Figure 16-9: Perspective Plan View of Grade Variability



Source SRK 2016

Figure 16-10: X-Section View of Grade Variability

16.3.4 Phase Design

The ultimate pit design has been separated into eight mine phases for sequenced extraction in the SRK production schedule. The design parameters for each phase are the same as those used for the ultimate pit design including assumed ramp widths. Mine phase designs were constructed by splitting the ultimate pit into smaller and more manageable pieces, while still ensuring each bench within each phase has ramp access. The phases were developed by balancing mining constraints with the optimum extraction sequence suggested by pit optimization results presented previously in Section 16.22.

The major design considerations include:

- Phases 1, 2 and 3 are low strip ratio phases for early ore exposure that can be mined independently, keep existing surface water drainages exposed and allow reasonable mining faces to accelerate bench sinking rate during pit pioneering;
- Phase 4 to the west of the pit has a high (relative) ratio of hard rock to saprolite rock. The 0hard rock is required for pre-production earthworks, foundations and road sheeting. For this reason, Phase 4 is mined before the more economic phases. Phase 4 also provides access to the best grade found in Phase 5;
- Phase 5 is to final on the west side of deposit and contains the highest grade ore during the payback period of the operation;
- Phases 6, 7 and 8 are continuations of the phase extraction moving to the east of the pit;
- Attempt to keep the minimum mining width at approximately 50 to 100 m; and
- The phases and direction of extraction allow for multiple benches on multiple elevations with a sump always available for pit dewatering. This means that during periods of heavy rainfall, perched benches will be available for extraction.

Phases 1 through 5 are important for mine operations as they will provide the mine with valuable practical experience on dealing with the saprolite, and the failed saprolite from previous natural historical slips;

Once the phases have been designed, solid triangulations are created for each phase as it cuts into topography from previous phases. These solid phases are then shelled (cut) on a 5 m lift height that corresponds to one block model block. These shells form a bench within each phase and represent the basic unit that is scheduled for the LoM production plan.

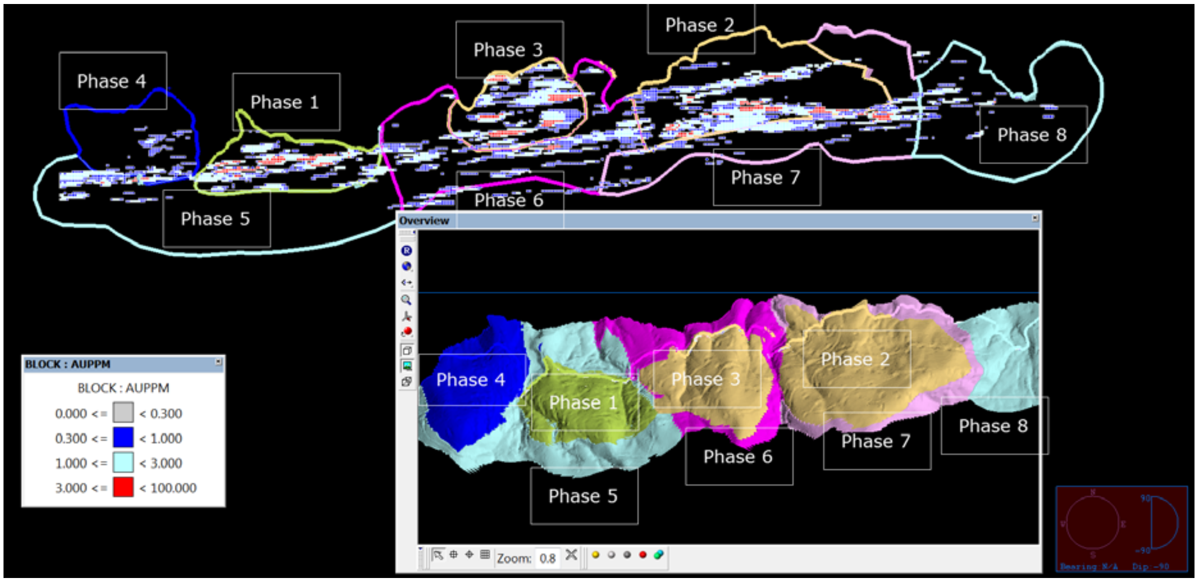
Table 16-10 details the phase inventory available for the pre-production and LoM schedule.

Table 16-10: Phase Inventory Available for Production Schedule

Phase		1	2	3	4	5	6	7	8	Total
Ore Tonnes	t	2,271,884	7,409,446	3,936,645	1,545,364	10,282,444	11,787,730	12,594,859	4,285,246	54,113,619
Waste Tonnes	t	6,595,576	20,828,069	7,553,556	11,737,243	67,414,297	47,282,013	50,790,647	29,551,968	241,753,368
Au Grade	g/t	2.01	1.62	1.50	1.41	1.78	1.44	1.50	1.53	1.58
Strip Ratio	w:o	2.90	2.81	1.92	7.60	6.56	4.01	4.03	6.90	4.47
Hard/Sap	h:s	1.18	1.70	1.09	0.96	5.72	7.33	9.95	4.33	4.23
Au Ounces	oz	146,866	385,522	189,307	70,165	589,062	545,702	607,887	210,885	2,745,396

Source: SRK, 2017

Figure 16-11 shows a graphical representation of the eight phases that were constructed for the LoM production schedule inventory.



Source: SRK, 2016

Figure 16-11: Phase Layout

16.3.5 Production Schedule

The mine production schedule utilized the CPLEX optimization tool within the Maptek™ Chronos scheduling package. Benches within each phase have a precedence relationship assigned to ensure top down mining in an orderly sequence. The objective function of the optimization was to maximize a simplified NPV calculation in each period, but still maintain a reasonable mining fleet. Optimizations were conducted on a monthly basis for 24 months after pre-production, followed by quarterly periods through the end of the mine life. The final scheduling restrictions were based on involved trial and error to prevent major production fluctuations as the pit progressed through the deposit.

The Project is located in a tropical environment that generally has two wet seasons, one dry season and transition periods where rain can be either high or low depending on the year. The SRK production schedule has modified the annual average target production rate and applied a 15% adjustment to the wet and dry seasons to account for mine performance. The 30% adjustment in production rate results in a mining rate of 68 kt/d in the wet season and 92 kt/d in the dry season, with the remainder of transition months targeting 80 kt/d. The restrictions were applied on a monthly and quarterly basis as defined in the production schedule.

Table 16-11 details the average monthly rainfall and correction factors applied by SRK.

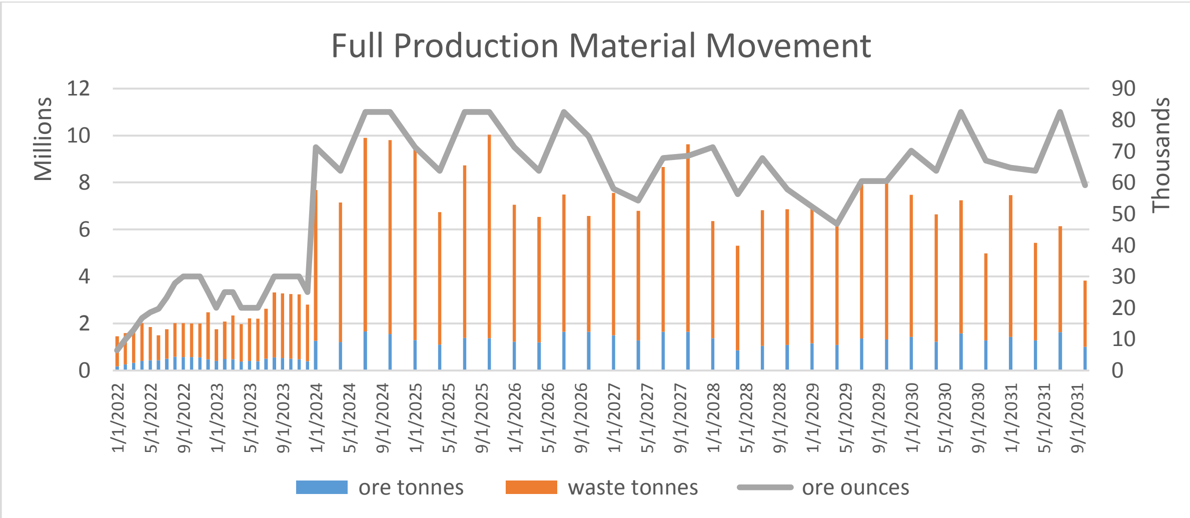
Table 16-11: Monthly Rainfall and Mining Rate Correction Factors

Deviation From Average	195.62	38%	13%	8%	47%	91%	40%	-13%	-44%	-77%	-76%	-50%	23%	
		-15	0	0	-15	-15	-15	0	15	15	15	15	0	
Monthly Target	80,000	68,000	80,000	80,000	68,000	68,000	68,000	80,000	92,000	92,000	92,000	92,000	80,000	

Year	January	February	March	April	May	June	July	August	September	October	November	December	Total
Mean Rainfall	270.09	220.90	211.72	286.61	373.58	274.61	170.15	109.48	45.68	46.14	97.88	240.66	2,628.86
Minimum Rainfall	0.00	0.00	0.00	2.12	0.00	0.00	0.00	0.00	1.25	0.42	7.54	62.18	1,200.74
Maximum Rainfall	753.18	692.28	587.42	615.68	706.52	436.05	382.89	415.96	175.01	165.37	296.22	469.14	3,439.20
Std. Dev. (σ)	176.26	172.28	132.92	173.64	160.89	103.37	78.66	69.23	36.65	44.75	65.34	100.84	530.01
Count	34	35	37	34	34	33	37	36	32	33	33	31	23
Departure From Average	38%	13%	8%	47%	91%	40%	-13%	-44%	-77%	-76%	-50%	23%	
Productivity Pass													
Productivity Adjustment	-15	0	0	-15	-15	-15	0	15	15	15	15	0	
Average Production Target	68,000	80,000	80,000	68,000	68,000	68,000	80,000	92,000	92,000	92,000	92,000	80,000	
	Wet Season												
	Average Target For Productivities												
	Dry Season												

Source: SRK, 2016

Figure 16-12 graphically illustrates the period variations in total ore and waste tonnes mined on a period basis during full mine production. The monthly and quarterly schedules form the basis of the economic model and ensure ore is exposed at all times during the LoM, with no extended periods of waste mining that can be hidden in an annual schedule.



Source: SRK, 2016

Figure 16-12: Full Production Period Schedule

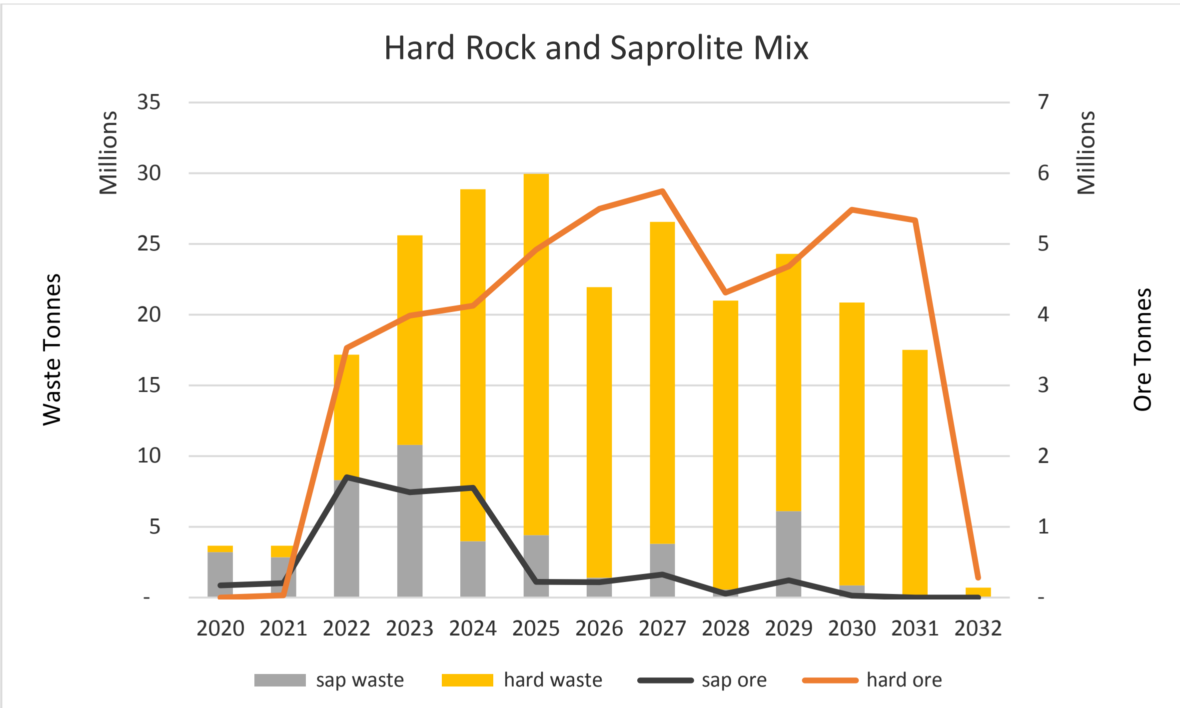
The annual mine production schedule for material coming directly from the pit is detailed in Table 16-12.

Table 16-12: Annual Production Schedule

	Units	Total	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2031
Ore Tonnes	t	54,113,619	172,825	235,205	5,230,741	5,470,017	5,673,310	5,134,778	5,713,459	6,071,250	4,363,816	4,923,913	5,508,089	5,335,140	281,078
Waste Tonnes	t	241,753,368	3,650,000	3,660,000	17,161,454	25,613,333	28,860,310	29,956,630	21,930,657	26,559,966	21,000,000	24,300,000	20,856,946	17,508,063	696,008
Ore Ounces	oz	2,745,396	4,241	7,726	250,448	300,000	300,000	300,000	292,293	248,588	253,158	220,000	283,373	270,187	15,381
Ore Au Grade	g/t	1.58	0.76	1.02	1.49	1.71	1.64	1.82	1.59	1.27	1.80	1.39	1.60	1.58	1.70
Strip Ratio	w:o	4.47	21.12	15.56	3.28	4.68	5.09	5.83	3.84	4.37	4.81	4.94	3.79	3.28	2.48
phase_1_bench	5 m	30	6.00	1.00	1.76	17.87	3.37	-	-	-	-	-	-	-	-
phase_2_bench	5 m	45	9.12	6.31	16.49	7.29	5.79	-	-	-	-	-	-	-	-
phase_3_bench	5 m	40	-	7.00	8.00	8.00	15.67	1.33	-	-	-	-	-	-	-
phase_4_bench	5 m	40	14.00	3.00	6.71	13.97	1.32	1.00	-	-	-	-	-	-	-
phase_5_bench	5 m	58	-	-	10.00	7.97	13.69	11.91	12.29	-	2.14	-	-	-	-
phase_6_bench	5 m	73	-	-	-	9.00	-	8.00	16.86	16.07	19.69	1.16	0.22	2.00	-
phase_7_bench	5 m	72	-	-	-	-	3.17	15.45	-	4.70	10.75	10.62	8.98	12.32	6.00
phase_8_bench	5 m	42	-	-	-	-	-	-	-	-	-	14.17	8.21	17.76	1.86
Sap Ore	t	6,193,006	172,825	202,631	1,700,817	1,487,015	1,549,955	219,157	215,951	324,451	51,747	242,997	25,460	-	-
Hard Ore	t	47,920,612	-	32,574	3,529,924	3,983,001	4,123,355	4,915,621	5,497,508	5,746,799	4,312,069	4,680,915	5,482,628	5,335,140	281,078
Sap Waste	t	46,225,474	3,217,402	2,853,594	8,301,442	10,781,996	3,966,641	4,407,434	1,407,141	3,786,499	552,985	6,103,460	846,880	-	-
Hard Waste	t	195,527,894	432,598	806,406	8,860,012	14,831,337	24,893,669	25,549,197	20,523,516	22,773,467	20,447,015	18,196,540	20,010,066	17,508,063	696,008
Total Tonnes	t	295,866,987	3,822,825	3,895,205	22,392,194	31,083,350	34,533,621	35,091,408	27,644,116	32,631,216	25,363,816	29,223,913	26,365,034	22,843,203	977,085
Hard Ore Cycle Time	mins	14	-	11	14	10	9	8	12	10	14	13	16	19	20
Hard Ore Distance	m		-	1,650	2,294	1,650	1,476	1,198	1,880	1,749	2,316	2,259	2,770	3,311	3,421
Hard Waste Cycle Time	mins	16	13	18	17	11	11	15	14	14	17	10	12	18	23
Hard Waste Distance	m		2,073	2,769	2,614	1,787	1,698	2,731	2,328	2,564	2,793	1,601	1,891	2,837	3,635
Sap Ore Cycle Time	mins	13	14	15	15	11	10	7	10	8	16	16	15	-	-
Sap Ore Distance	m		1,983	2,024	2,257	1,508	1,595	1,004	1,891	1,485	2,997	2,988	2,708	-	-
Sap Waste Cycle Time	mins	14	16	19	18	14	11	14	9	12	11	11	10	-	-
Sap Waste Distance	M		2,441	2,627	2,626	2,187	1,683	2,584	1,654	2,323	1,989	1,835	1,794	-	-

Source: SRK, 2016

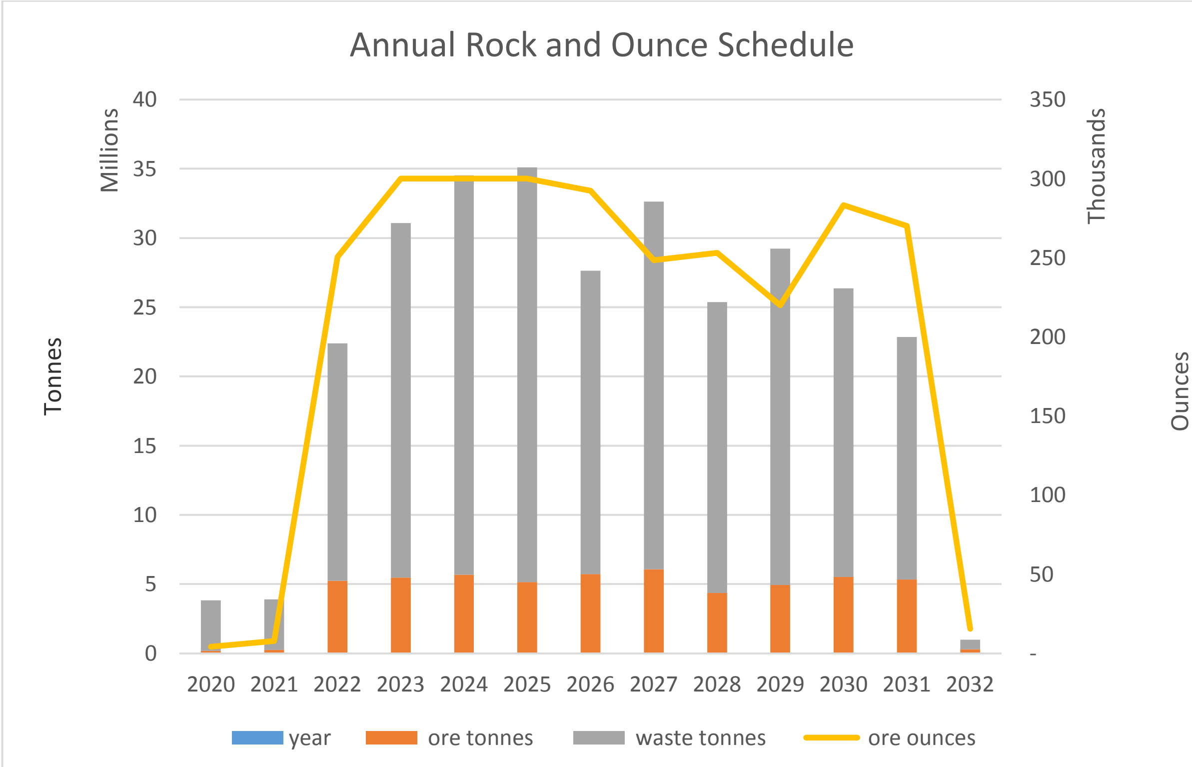
Figure 16-13 illustrates the annual production showing the split of hard rock and saprolite, ore and waste schedule. The pre-stripping targets the removal of saprolite, so that hard rock and introduction of 91 t trucks was possible when full mining operations commence (2022 or Year 1). The ratio of saprolite for both ore and waste is high through 2023 as the initial high grade starter pits are mined along pit strike. As the pit deepens the saprolite is mined in smaller quantities as phase pushbacks move up the side of the hill to final design. The increase of saprolite waste in year 2029 is a result of mining Phase 8 to the extremity of the pit on the eastern flank.



Source: SRK, 2016

Figure 16-13: Hard Rock and Saprolite Ore and Waste Schedule

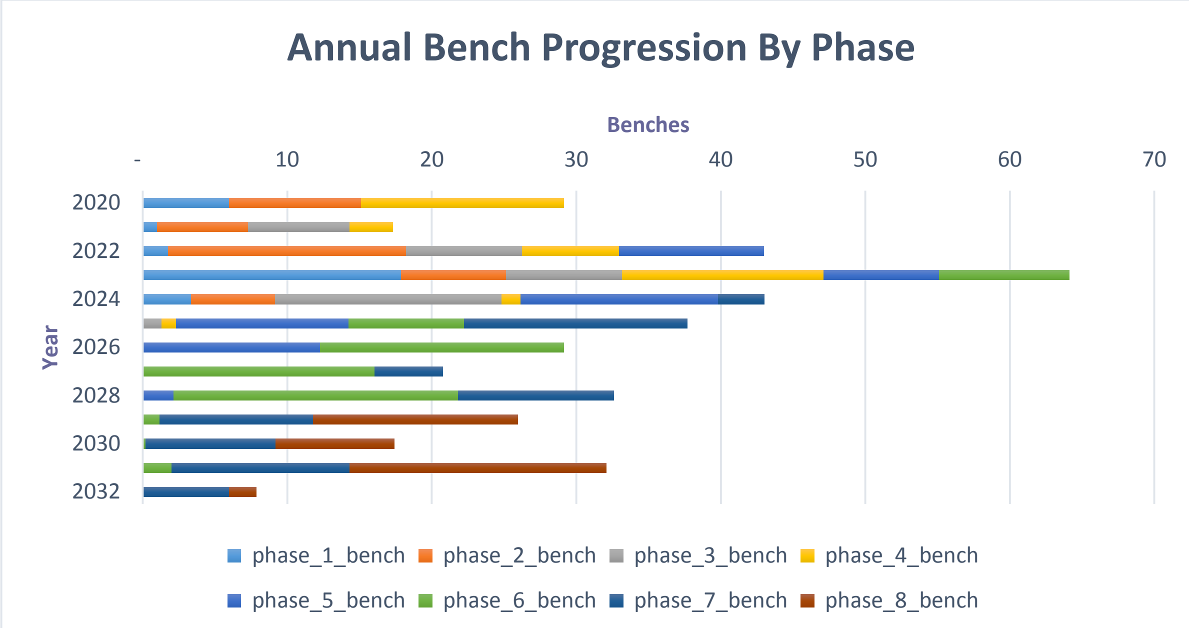
Figure 16-14 illustrates the annual ore and waste schedule. Generally, the west portion of the ultimate pit mined first is higher grade areas, thus yielding 300 koz Au per year. As the mine moves to the east, a lower grade section in the middle of the deposit must be mined and the ounces reduces until the eastern ore at depth is exposed at the end of the mine life and gold production increases. Ore is mined over the 4.6 Mt/y ore processing limit to ensure marginal material (low-grade) does not displace higher grade ore in the schedule.



Source: SRK, 2016

Figure 16-14: Saprolite, Hard Rock and RoM Production Schedule

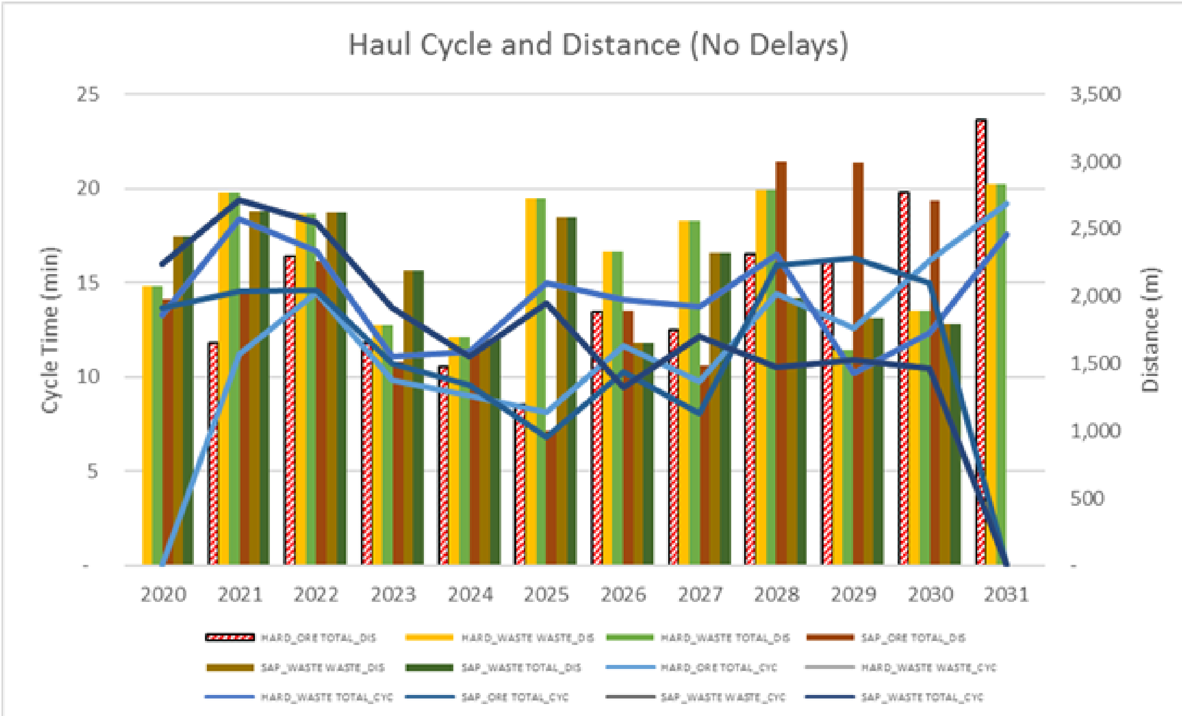
The bench sinking rate for the production was set at two benches per month which is aggressive, but is offset because the small bench height (5 m), and the ore benches are reasonably big allowing partial excavation and terracing. The smaller benches mined as part of pioneering operations are generally waste, and sinking rates may be considerably more than two benches per month when access is established. The highest bench sinking rate occurs in year 2024, but it is also the time when the majority of phases are available for mining. Even with stripping brought forward in 2024, there is a drop in available mill feed in year 2026 as the pit moves through the central core of lower grade material. The multiple benches mined from each phase in each year will provide good operational flexibility for the mine. Figure 16-15 illustrates the annual bench sinking rates.



Source: SRK 2016

Figure 16-15: Bench Sinking Rate

The haul cycle times are flagged into the SRK mine block model and reported on an annual basis. Each group of phases with waste destined for a particular dump are grouped into sectors to ensure the correct cycle time is applied. The cycle times are high in the beginning of the mine life when pioneering and benches are at higher elevations. As the mine lowers and dump elevation increases, the haul cycle times are at their lowest when the highest rates of production are required. At the end of the mine life the pit depth and dump height increases. The waste cycle times are higher than normal in 2022, 2025, 2027 and 2028 due to the extended haul required to supply bulk material to the tailings embankments. Generally, the result of the cycle time analysis suggests the trucks will be averaging 20 km/h, which is reasonable given the terrain and potentially slippery conditions that may be found during operations. Figure 16-16 illustrates the instantaneous annual haul distance and cycle times used as the basis for fleet estimation.



Source: SRK 2016

Figure 16-16: Instantaneous Cycle Times and Haul Distance

The Project mine production schedule as described above has noted the years of 2020 and 2021 as the pre-production mining years (Years -2 and -1, respectively). Production mining occurs from 2022 through 2031 (Years, 1 through 10), with minor mining operations occurring in 2032. Stockpile re-handling takes place mainly in Year 11 (2032) and Year 12 (2033).

16.3.6 Waste Rock Storage

Overburden materials were sent to two WRD locations near the pit rim to reduce haul times for mine haulage trucks, and were scheduled according to where the material was exiting the pit. In the first five years, waste was scheduled to the WWRD and the starter CWRD both north of the pit. As the WRDs were comprised of two phases of construction for the WWRD (North and South) and four phases for the CWRD (Central, West, North and East), the progression of advancement was generally west to east following the pit progression.

The total storage and volumetric capacity of the WRDs are detailed in Table 16-13 below.

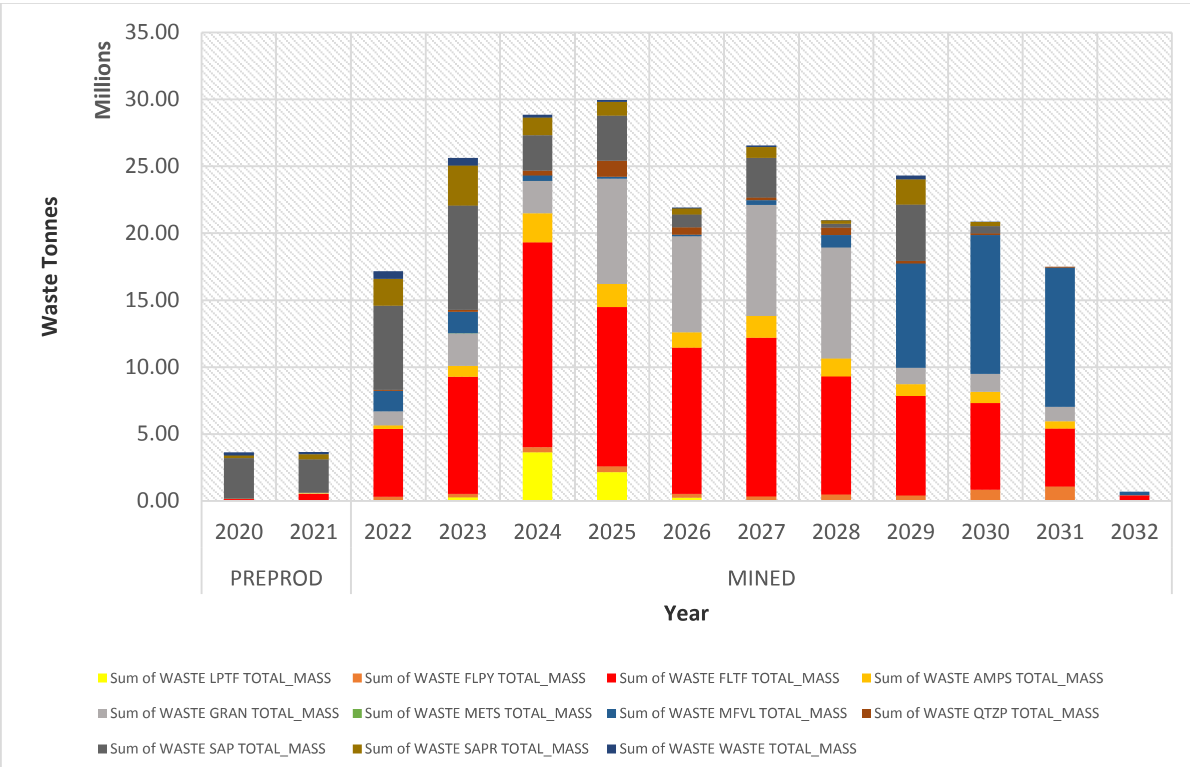
Table 16-13: Overburden Storage Areas

Dump	Tonnes	Volume m^3	Year Started	Year Complete
West Waste RockDump	54,255,080	27,127,540	2020	2025
Central Starter Waste Rock Dump	31,815,311	15,907,655	2020	2024
Central Waste Rock Dump	156,641,953	78,320,977	2024	2031
Grand Total	**242,712,344**	**121,356,172**	**2020**	**2031**

Source: SRK, 2016

Because the WRDs are staggered, progressively rehabilitation of the waste slopes will be possible for the West Dump after Year 5. Only rehabilitation of partial slopes for the Central Dump will be possible prior to the end of mining.

Figure 16-17 details the amount of PAG material as a function of NAG waste that is mined during operations. Generally, the PAG materials are the mineralized tuffs that are coloured in red, orange and yellow. Because there is a lot of saprolite (NAG) material in the early years of the operation, it is reasonable to assume the saprolite will act as a buffer for PAG material placed in the WWRD (with the majority of PAG material).



Source: SRK, 2016

Figure 16-17: Annual NAG/PAG Schedule

16.3.7 Haulage Profile

A significant mine design requirement for the production schedule was to use a reasonably consistently sized loading and haulage fleet that would be able to supply the mill with the required ore feed, and to not become waste bound. In order to do this, the cycle time and one way distance had to be estimated into the block model so the different haulage lengths from different parts of the pit could be accurately calculated.

Table 16-14 details the speed with which the trucks have been estimated to run at different gradients. This is calculated for both loaded and empty portions of the haulage cycles. The speed has been capped at 30 km/h for safety reasons and to not bias the operating speed of the trucks. Loaded truck speeds are capped at 25 km/h.

Table 16-14: Rimpull Curve Representing Truck Speeds by Gradient for 91 t Trucks

Truck 91 t	Gradient %	Speed Uphill km/h	Speed Downhill km/h
Loaded	Flat	25	25
	0	25	25
	2	18.5	25
	4	17	25
	6	14.9	21
	8	11.5	19
	10	10.4	17
	15	5	5
Empty	Flat	30	30
	0	30	30
	2	30	30
	4	27	30
	6	23	24
	8	20	18
	10	17	15
	15	10	10

Source: SRK, 2016

Haulage times were based on material from each phase going to either the CWRD or WWRD or crusher location. Because there are multiple ramps within the pit, the haulage profile within the pit to an exit point was estimated with an 8% grade, and the in-pit time was then added to the time taken defined by haul road strings to the WRD or crusher. The cycle time and distance was stored in the block model and the haul cycles were matched with a routing block code in the block model. For example, waste blocks are assigned the waste cycle time based on the distance of the block to the WRD. Strings to dumps were based on centroid locations. In the years where bulk fill is required for the tailings embankment construction, the waste haul routes were extended to account for the additional haul times to supply waste rock at the required rate. Individual locations were not specified, but the nature of the tailings embankment construction will result in minimal elevation changes.

Haulage times reported in the block model were instantaneous times and do not have any efficiency corrections (other than speed around corners) delays, spot times or dump times. These corrections were made in the SRK truck fleet estimation spreadsheet that used the cycle times reported as the basis for the estimate.

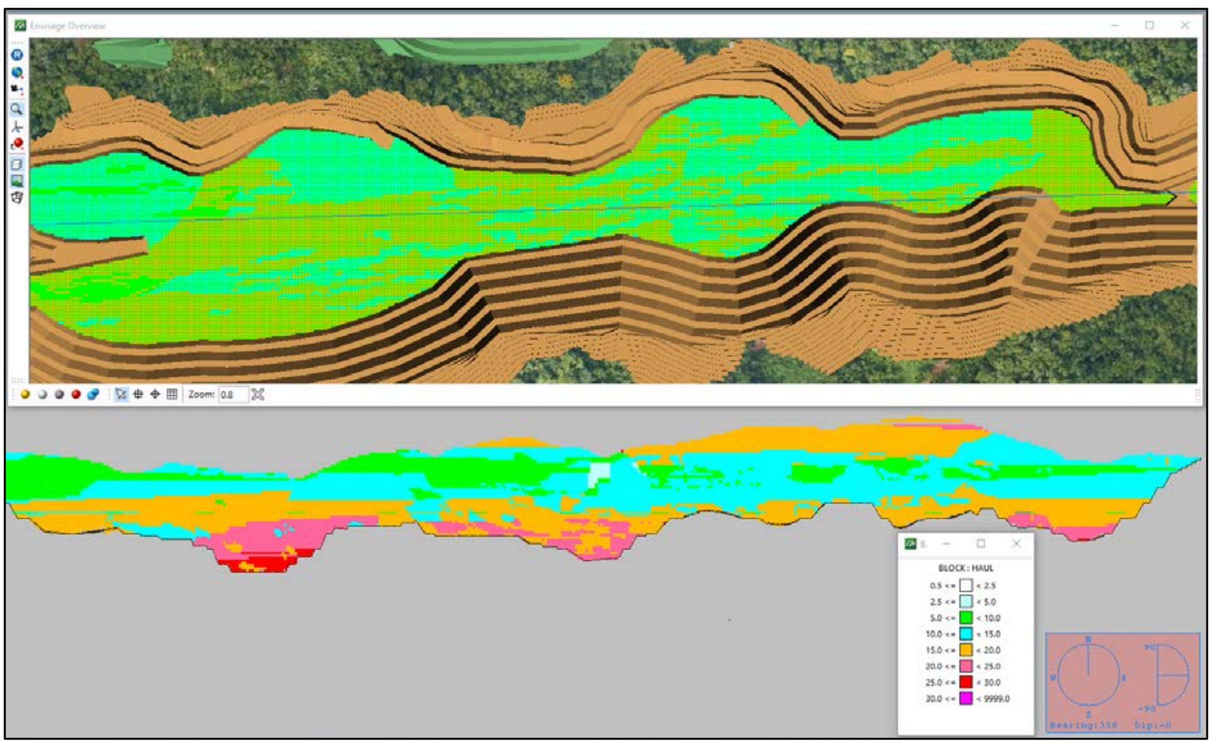
Table 16-15 details the cycle times broken down by year the mill feed and waste is to be mined.

Table 16-15: Instantaneous Cycle Time and Distance for Basis of Fleet Estimate

Period	Saprolite				Waste Rock			
	Cycle Time min*		One Way Distances m		Cycle Time min*		One Way Distances m	
	Waste	Ore	Waste	Ore	Waste	Ore	Waste	Ore
2020	16.0	13.7	2,441	1,983	13.3	11.2	2,073	1,650
2021	19.4	14.5	2,627	2,024	18.4	11.2	2,769	1,650
2022	18.2	14.6	2,626	2,257	16.7	14.5	2,614	2,294
2023	13.6	10.7	2,187	1,508	11.1	9.9	1,787	1,650
2024	11.1	9.6	1,683	1,595	11.4	9.0	1,698	1,476
2025	13.9	6.8	2,584	1,004	15.0	8.1	2,731	1,198
2026	9.4	10.3	1,654	1,891	14.1	11.6	2,328	1,880
2027	12.2	8.0	2,323	1,485	13.7	9.8	2,564	1,749
2028	10.5	16.0	1,989	2,997	16.5	14.4	2,793	2,316
2029	10.9	16.3	1,835	2,988	10.2	12.6	1,601	2,259
2030	10.5	15.0	1,794	2,708	12.3	16.2	1,891	2,770
2031	N/A	N/A	N/A	N/A	17.6	19.2	2,837	3,311
2032	N/A	N/A	N/A	N/A	22.9	19.6	3,635	3,421
2033	N/A	N/A	N/A	N/A	N/A	N/A	N/A	N/A

Source: SRK, 2016
Note: *Truck cycle moving times with 100% hauling efficiency (unadjusted).
This does not include truck spotting, loading and dumping times.

Figure 16-18 illustrates the haulage cycles times that have been coded into the block model and reported as part of the production schedule.



Source: SRK, 2016

Figure 16-18: Cross-Section of Haulage Cycle Time

16.4 Mining Operations

16.4.1 General Requirements and Fleet Selection

Mining activities will include site clearing, removal of growth medium (topsoil), free-digging, drilling, blasting, loading, hauling and mining support activities. Material within the pit will be generally blasted on a 5 m high bench. Most of the saprolite material (approximately 18% of the total material to be mined) can be loaded directly with hydraulic excavators without the need for blasting. WRDs will be used for material below the CoG and stockpiles for lower-grade ore above the CoG. Most ore will be sent directly to the primary crusher. The stripped waste material will be placed in dumps to the north of the pit, and lower-grade ore placed in a stockpile, near to the primary crusher location.

Because of the large amount of rainfall, hilly terrain, and amount of saprolite, SRK developed a mixed fleet mining. The first fleet was comprised of 6.7 m^3 capacity excavators loading 40 t ADTs, and will be used for pioneering excavation, most of the saprolite mining and can also assist with selective ore mining. The ADTs can work in tight terrain and have six-wheel drive system which is better suited for navigating slippery saprolite road and bench conditions. As the larger proportion of saprolite is removed and drainage improved, the second larger mining fleet of 12.0 m^3 capacity excavators and 91 t capacity rear dump trucks will perform the majority of the bulk production. The site mining roads will require road surfacing material up to 1 m thick.

Specific requirements dictated the selection of mining equipment types and sizes. The fleet of 12.0 m^3 capacity hydraulic excavators will be primarily used for loading waste and ore in the open pits and a 12.3 m^3 capacity FEL for flexibility with loading pit waste. A smaller 6.4 m^3 capacity FEL will be used with the 40 t ADTs for re-handling ore from the low-grade stockpile. Trucks have been matched to the loading equipment units. Additional equipment units were provisioned when required, in keeping with the planned mine production schedule requirements.

The major mine equipment fleet requirements were based on the annual mine production schedule, the mine work schedule, and shift production estimates. The fleet estimate generated by SRK was based on an internal spreadsheet, and equipment consumption information for fuel, lube, tires, etc., was based on Caterpillar Performance Handbook Edition 45 (Caterpillar, 2015) information for the particular class of equipment, and is not company specific. The mine equipment requirements and costing were based on the purchase of new equipment. The equipment fleet selection and requirements are further discussed in the individual sections that follow in this report.

It was planned that all mine mobile equipment would be diesel-powered, in order to avoid the requirement to provide electrical power into the pit working areas. This will negate the need for overhead electrical lines and electrical trailing cables, which would pose operational complications with the mining operations.

The mine operations schedule is proposed to include two 12-hour shifts per day, seven days per week for 355 days per year, which includes an annual allowance of 10 days downtime for weather delays for most of the mine operations, and 15 days downtime for weather delays for the drilling operations. Mine productivity rates and costs involved estimation of the productive operating time per 12-hour shift. Non-productive time per shift includes shift change (travel time), equipment inspections, fueling and operator breaks. It was estimated that the total time per shift for these items will be 1.8 hours.

A blasting contractor will perform the majority of blasting related operations. Blasting will normally be conducted at lunchtime breaks and will not represent an additional delay to the operations. The scheduled production time (scheduled operating hours) was therefore estimated at 10.2 hours per shift, representing a (shift) utilization of 85% of the 12-hour shift period (and excludes mechanical availability and work efficiency factors).

In addition, allowances were made for work efficiencies including equipment moves (production delays while moving to other mining areas within the pit), and certain dynamic operational inefficiencies. These work efficiencies are further discussed in the respective sections for drilling, loading and hauling.

Major mining equipment mechanical availabilities were estimated at 80% for drills, 80% for loading equipment, and 85% for haul trucks. In Year 4, haul truck availabilities were allowed to be increased to 88% as part of a specifically planned program to improve availabilities in that year, in order to avoid purchase of additional equipment units that would only be needed for that year.

Table 16-16 shows the mining equipment requirements by year for the LoM mine plan.

The Project mining production schedule has previously described the years of 2020 and 2021 as the pre-production mining years. Production mining occurs after this from 2022 through 2031, with minor mining operations occurring in 2032. These years have been described alternatively in this section, for pre-production as Year -2 (2020) and Year -1 (2021), and for production mining as Year 1 (2022) through Year 10 (2031), with minor mining operations occurring in Year 11 (2032). Low-grade ore stockpile re-handling takes place mainly in Year 11 (2032) and Year 12 (2033).

Table 16-16: Planned Mining Equipment Fleet

Equipment Units	Make	Model	Size	Yr -2	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12
Drilling																	
Blasthole drill	Atlas Copco	SROC D65	152 mm	1	2	3	4	4	4	4	4	4	4	4	3	1	-
Loading																	
FEL	Komatsu	WA600-8	6.4 m^3	1	1	1	1	1	1	1	1	1	1	1	1	1	1
FEL	Komatsu	WA800-3EO	12.3 m^3	-	-	1	1	1	1	1	1	1	1	1	1	1	-
Hydraulic excav	Komatsu	PC1250LC-8	6.7 m^3	2	2	2	2	2	2	2	2	2	2	1	1	-	-
Hydraulic excav	Komatsu	PC2000-8	12.0 m^3	-	-	3	3	3	3	3	3	3	3	3	3	2	2
Hauling																	
Haul truck – ADT	Komatsu	HM400-5	40 t	8	8	9	9	9	9	3	6	5	5	4	4	-	-
Haul truck – Rigid	Komatsu	785-7	91 t	-	-	13	14	15	17	17	17	17	17	17	17	4	4
Other Mine Equip																	
Crush/Screen Plant	Manufacturer	Jaw/Cone	335 kW	1	1	1	1	1	1	1	1	1	1	1	1	-	-
Track dozer	Caterpillar	D10T	447 kW	4	4	4	4	4	4	4	4	4	4	4	3	1	1
Wheel dozer	Caterpillar	834H	372 kW	-	-	1	2	2	2	2	2	2	2	2	1	1	1
Motor grader	Komatsu	GD675-6	165 kW	3	3	4	4	4	4	4	4	4	4	4	3	2	2
Backhoe loader	Caterpillar	450E	102 kW	1	1	1	1	1	1	1	1	1	1	1	1	-	-
Water truck	Scania	P410CB 8X4	30,000L	2	2	3	3	3	3	3	3	3	3	3	3	3	1
Excavator	Komatsu	PC800LC-8	363 kW	2	2	2	2	2	2	2	2	2	2	2	2	1	1
Compactor	Caterpillar	CS/CP 533E	97 kW	2	2	2	2	2	2	2	2	2	2	2	1	1	1
Support Equip																	
Low bed trailer	Manufacturer	Model	360 t	-	-	1	1	1	1	1	1	1	1	1	1	-	-
Truck crane	Manufacturer	Model	120t	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Recovery truck	Scania	G460CB 8X8	360 kW	-	-	1	1	1	1	1	1	1	1	1	1	-	-
Fuel truck	Scania	P410CB 8X4	30,000L	1	1	2	2	2	2	2	2	2	2	2	2	1	1
Lube ruck	Scania	P410CB 8X4		1	1	2	2	2	2	2	2	2	2	2	2	1	1
HD mech truck	Scania	P360CB 6X4		1	1	2	2	2	3	3	3	3	3	3	2	1	1
Welding truck	Scania	P360CB 6X4		1	1	1	1	1	1	1	1	1	1	1	1	1	1
Tire service truck	Scania	P360CB 6X4		1	1	2	2	2	2	2	2	2	2	2	2	1	1
Flatbed truck	Scania	P360CB 6X4	19 t crane	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Personnel van/bus	Manufacturer	Model		1	1	2	2	2	2	2	2	2	2	2	2	1	1
Service pickup	Manufacturer	4x4		10	10	20	20	20	20	20	20	20	20	20	20	15	15
Light plant	Manufacturer	Portable	8 kW	10	10	15	15	15	20	20	20	20	20	20	20	15	10
Stemming truck	Scania	P360CB 6X4	Owner	-	1	1	1	1	1	1	1	1	1	1	1	1	1
Pit pumps/gen	Surface Water Equip. List		Owner	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Blast Contractor																	
Flatbed truck	Manufacturer	Model	Contract	1	1	1	1	1	1	1	1	1	1	1	1	-	-
ANFO/Emuls. Truck	Manufacturer	15 t	Contract	1	1	1	1	1	1	1	1	1	1	1	1	-	-
Blasters crew truck	Manufacturer	4x4	Contract	1	1	1	1	1	1	1	1	1	1	1	1	-	-

Source: SRK, 2017

16.4.2 Drilling

The planned drilling equipment fleet consisted of Atlas Copco SROC D65 units, or equivalent units. This fleet was based on drilling 152 mm blastholes to an average depth of 5.75 m (including a 0.75 m sub-drill) for development of 5 m high benches. The drills can single-pass drill (no rod changes) such holes.

The planned nominal production blasthole pattern is equivalent to a 5.3 m x 5.3 m pattern (spacing and burden) in waste and 5.1 m x 5.1 m square pattern in ore, however, in practice the burden and spacing will vary. For the main production drilling an average instantaneous drilling rate of 0.6 m/minute was estimated for waste and ore. Allowances were made in the estimate of drilling requirements for wall control holes, some locally closer spaced drilling and re-drills, equivalent to drilling an additional 10% of holes (as an added contingency factor above the basic production drilling estimate). Allowance was also made in drilling productivity for drilling setups and moving to new working areas.

Approximately 85% of the designated saprolite material will be able to be freely excavated without blasting. Less than 15% of the designated saprolite material will be saprock and will require blasting. However, saprolite ore and some saprolite waste will need to be drilled for grade control purposes. Drill fleet requirements were based on drilling all of hard rock within the planned open pits (82% of all material), all of the saprolite ore, and 50% of the saprolite waste.

The details of the drilling and blasting assumptions are displayed in Table 16-17.

Table 16-17: Blasthole Estimation per Drill Unit

Parameter	Units	Waste	Ore
Instantaneous Penetration Rate	m/h	36	36
Hole Diameter	mm	152	152
Hole Area	m^2	0.015	0.015
Bench Height	m	5.00	5.00
Sub-drill Height	m	0.75	0.75
Total Hole Length	m	5.75	5.75
Loaded Length	m	4.31	4.31
Stemming Height	m	1.44	1.44
Loaded Hole Volume	m^3	0.078	0.078
Explosive Density	kg/m^3	1.15	1.15
Explosive per Hole	kg	90	90
Powder Factor	kg/t	0.22	0.24
Tonnage Blasted per Hole	t/hole	409	375
Volume Blasted per Hole	m^3/hole	143	129
Square Pattern Spacing	m	5.34	5.08
Drilling Time per Hole	min/hole	9.6	9.6
Non-Productive Time per Op. Hr.	min	10.0	10.0
Tram and Setup Time per Op. Hr.	min	9.5	9.5
Drilling Time per Hour	min	40.5	40.5
Holes Drilled per Op. Hour	holes	4.23	4.23
Length Drilled per Op. Hour	m/op hr	24.3	24.3
Drill Productivity w/out Re-drills	t/op hr	1,728	1,584
Drill Productivity w/ 10% Contingency	t/op hr	1,555	1,426

Source: SRK, 2017

Table 16-18 shows selected drilling statistics based on the planned drilling equipment and drilling patterns for waste and ore.

Table 16-18: Selected Drilling Statistics per Drill Unit

Rock Type	Waste/Ore Pattern Size m x m	Drilling Tram and Set Up Time min/op hr	Drilling Penetration Rate m/min	Drilling Time per Blasthole min	Moving and Delay Time min/op hr	Production per Unit 100% Available) t/op hr*
Waste	Sq. 5.34 x 5.34	9.5	0.6	9.6	10.0	1,555
Ore	Sq. 5.08 x 5.08	9.5	0.6	9.6	10.0	1,426

Source: SRK, 2017
Note: * Includes contingency allowance of 10%.

Table 16-19 shows selected drilling productivity information based on the planned drilling equipment for waste and ore. Annual production capacity per drill is 8.9 Mt/y for waste and 8.1 Mt/y for ore.

Table 16-19: Drilling Productivity per Drill Unit

Rock Type	Production per Unit (100% Available) t/op hr	Planned Operating Hours per Shift scheduled op hrs	Planned Operating Hrs per Year * scheduled op hrs	Estimated Mechanical Availability ** %	Actual Operating Hours per Year op hrs	Annual Production Capacity per Unit Mt/y
Waste	1,555	10.2	7,242	80%	5,712	8.9
Ore	1,426	10.2	7,242	80%	5,712	8.1

Source: SRK, 2017
Note: * Includes allowance of 15 days downtime for weather delays.
** Typical mechanical availabilities for drills used.

16.4.3 Blasting

An explosives provider for the mine will have explosives storage facilities at the mine site, located to the west of the MSA. The explosives provider will transport products to the Cayenne Port. From there, a transportation contractor will transport products to the mine site. During the pre-production period bulk blasting agent product (non-sensitized emulsion) will be supplied in 25 t isotanks. The transportation contractor will also transport blasting accessory products to the mine site, and deliver these to the explosives provider's storage facilities. Blasting accessories will be stored in suitable explosives magazines, which will separate detonators from other products. Earth berms will be constructed between storage structures as required to comply with explosive storage regulations.

The explosives provider for the mine will also be the blasting contractor for the mine. Commencing at the same time as the mill production (start of Year 1), the blasting contractor will start production of bulk emulsion using an emulsion plant located within the explosives storage facilities compound, which will be capable of sufficient bulk emulsion production over the life of the planned mining operations. (Orica Mining Services and AEL Mining Services were contacted to provide information for the BFS.)

The blasting contractor will have mobile equipment units including pickup trucks, forklift and an emulsion truck (approximately 15 t capacity) for delivery into blastholes. During peak production there will be two emulsion trucks in use. Delivery of bulk explosives to the blast sites will be during daylight hours. Blasting will normally be conducted at lunchtime breaks and will not represent an additional delay to the operations. The blasting contractor will manage and conduct the blasting operations. Blasting contractor staff will include managers, supervisors and operators for the blasting operations, and for the emulsion plant when that is in operation.

The mine owner will provide a stemming truck and operators dedicated to providing stemming material for the blastholes, which would consist of crushed rock or hard rock drill cuttings.

All saprock and hard rock within the open pit design is planned to be blasted. The powder factor for production blasting was estimated to be 0.22 kg/t (kg explosives per tonne of rock) for waste and 0.24 kg/t for ore.

16.4.4 Loading

Loading equipment selection included having a combination of small and large diesel-powered hydraulic excavators, and one small and one large FEL for operational flexibility. The hydraulic excavators are capable of mining ore more selectively, and will be used for mining all of the ore, and most of the waste. The small FEL (6.4 m^3 capacity) will be used for low-grade stockpile re-handling and, when required for more extended periods, RoM stockpile tramming duties. The large FEL (12.3 m^3 capacity) will be primarily used for loading waste.

The loading equipment fleet for the pre-production years (-2 and -1) of the mining operations was planned to consist of two smaller hydraulic excavators (6.7 m^3 capacity), and one FEL (6.4 m^3 capacity). These excavators will load a fleet of 40 t capacity ADTs placed in service mainly for pioneering, saprolite mining, and selective mining situations within the pit, and the FEL will be used mainly re-handling the low-grade stockpile ore to the primary crusher.

At the start of mill production (Year 1) an additional larger loading fleet will be added consisting of three hydraulic excavators (12.0 m^3 capacity), and one FEL (12.3 m^3 capacity). This equipment will load a fleet of 91 t capacity haul trucks.

The hydraulic excavators were estimated to be able to free-dig 85% of the designated saprolite waste and ore within the planned open pit. Designated saprolite material is approximately 18% of the total material within the planned open pit, and includes some saprock (less than 15% of all the saprolite material). Average dry density for saprolite waste was estimated to be 1.87 t/m^3, and saprolite ore 1.93 t/m^3. Average dry density for waste rock was estimated to be 2.87 t/m^3 and ore rock 2.91 t/m^3. Saprolite moisture content, for loading purposes, was estimated to be 30% on average (varying with season and depth), and swell in loading to be 20%. Hard rock moisture content was estimated to be 6% on average, and swell in loading to be 40% (Table 16-20).

Table 16-20: Estimated Excavator Loading Parameters

	Units	Waste Saprolite	Waste Rock	Waste Rock	Ore Rock
Excavator Model/Capacity		**PC1250-8 (6.7 m^3)**	**PC1250-8 (6.7 m^3)**	**PC2000-8 (12.0 m^3)**	**PC2000-8 (12.0 m^3)**
Truck Capacity		**40 t**	**40 t**	**91 t**	**91 t**
Loose Material Swell Factor	ratio	1.2	1.4	1.4	1.4
Material Moisture by Weight	%	30%	6%	6%	6%
Heaped Bucket Capacity	m^3	6.7	6.7	12	12
Bucket Fill Factor	%	74%	69%	70%	69%
Tonnes per Pass – Wet	wt	10.0	10.0	18.2	18.2
Truck Capacity – Dry	t	40	40	91	91
Truck Capacity	m^3	24	24	64	64
Number of Passes – Weight Basis	no.	4	4	5	5
Truck Capacity Utilized – Volume	%	83%	77%	66%	65%
Truck Capacity Utilized – Weight	%	100%	100%	100%	100%
Truck Load Time (Incl. Spotting)	min	2.55	2.55	3.34	3.59
Unit Moving and Clean-up per Hour	min	14.0	14.0	14.0	15.0
Trucks Loaded per Hour	no.	18	18	14	13
Maximum Loading Productivity – Dry	t/hr	557	683	1,184	1,079

Source: SRK, 2017

In most cases the haul trucks were assumed to be loaded up to 100% of their designed maximum weight capacity, allowing for the moisture content. The total truck loading times included a truck spotting (initial positioning of the trucks for loading) time of 42 seconds.

Allowance was made in the loading productivities for cleaning up working faces with the help of a wheel dozer, and for moving of the loading equipment to new working areas around the pits.

Table 16-21 shows selected loading productivity information based on the planned loading equipment for saprolite.

Table 16-21: Loading Productivities in Saprolite

Equipment Type/ Material Type	Production per Unit 100% Available dry t/op hr	Planned Operating Hrs per Shift scheduled op hrs	Planned Operating Hrs per Yr * scheduled op hrs	Estimated Mechanical Availability ** %	Actual Operating Hrs per Yr op hrs	Annual Ore Production Capacity per Unit dry Mt/y
6.7 m^3 Excav/Waste	557	10.2	7,242	80%	5,794	3.2
6.7 m^3 Excav/Ore	506	10.2	7,242	80%	5,794	2.9

Source: SRK, 2017
Note: * Includes allowance of 10 days downtime for weather delays.
** Typical mechanical availabilities for excavators.

Table 16-22 shows selected loading productivity information based on the planned loading equipment for waste rock.

Table 16-22: Loading Productivities by Unit Type in Waste Rock

Equipment Type/ Material Type	Production per Unit 100% Available dry t/op hr	Planned Operating Hrs per Shift scheduled op hrs	Planned Operating Hrs per Yr * scheduled op hrs	Estimated Mechanical Availability ** %	Actual Operating Hrs per Yr op hrs	Annual Ore Production Capacity per Unit dry Mt/y
6.7 m^3 Excav/Waste	683	10.2	7,242	80%	5,794	4.0
6.7 m^3 Excav/Ore	626	10.2	7,242	80%	5,794	3.6
12 m^3 Excav/Waste	1,184	10.2	7,242	80%	5,794	6.9
12 m^3 Excav/Ore	1,079	10.2	7,242	80%	5,794	6.3
12 m^3 Loader/Waste	1,333	10.2	7,242	80%	5,794	7.7

Source: SRK, 2017
Notes: * Includes allowance of 10 days downtime for weather delays.
** Typical mechanical availabilities for excavators and loaders used.

As part of the mining operations, an allowance was made for re-handling 2% of the plant ore feed from the RoM ore stockpile adjacent to the primary crusher during extended periods when weather events may cause interruptions to the mining operations. During normal operations a plant loader will feed the primary crusher from the RoM ore stockpile, as required. Re-handling of the low-grade stockpile ore has been included in the mining loading/hauling operations. Additional loading operations by the smaller FEL included crushed waste backfill to be hauled to the pits for road and ramp surfacing.

16.4.5 Hauling

The truck sizes selected were determined by loading unit/truck matching, maintaining the necessary degree of operational flexibility, and meeting production requirements. Waste will be hauled either to the WRDs or to the TSF embankments in particular years, including the pre-production years. Ore will be hauled either to the primary crusher or the low-grade ore stockpile.

The hauling equipment fleet for the pre-production years (-2 and -1) of the mining operations was planned to be comprised of 40 t capacity ADTs. This type of unit is commonly used in high rainfall saprolite mining conditions. A fleet of eight 40 t capacity ADTs was planned for the pre-production mining operations. The maximum fleet size will be nine units starting in Year 1.

The main production hauling equipment fleet for the mining operations, starting in Year 1, was planned to be comprised of 91 t capacity rear dump trucks. The initial fleet size will be 13 units, increasing to 17 units in Year 4, and maintained at that size until Year 10. Years 11 and 12 mainly involve re-handling haulage from the low-grade stockpile and will require four trucks at most.

Various haul profiles were developed for different time periods, and haulage cycle times from the pits were estimated for waste and primary crusher ore, which was considered to be the same as the low-grade stockpile ore for the purposes of haulage cycle analysis. In addition, waste haulage of saprolite and waste rock for use in construction of the TSF embankments in particular years was allowed for. Haulage cycle times were longer for a portion of the waste production in certain years. Base haul cycle times were estimated using Vulcan™ software, and which were subsequently factored (increased) for practical operational hauling aspects to reflect realistic cycle times.

The Maptek Vulcan™ haulage module was used to calculate the cycle times and distances. Lines were drawn from benches for each pit phase to the destinations. Block model blocks were then coded for cycle times and one way distances reported. Due to a mining operation with a mixed fleet, SRK standardized the speeds, since both truck types will be sharing the same ramp and road systems.

Table 16-23 shows the hauling efficiency factors used to adjust the base haul cycle times estimated from the Vulcan™ software. These efficiency factors increase the truck moving cycle times based on the one-way haul distance. (These factors were developed by Caterpillar Inc.).

Table 16-23: Haulage Truck Efficiency Factors

Minimum One-Way Haul Distance m	Maximum One-Way Haul Distance m	Haulage Efficiency Factor %
150	300	77%
300	600	80%
600	1,050	86%
1,050	1,500	90%
1,500	2,450	92%
2,450	Further	95%

Source: SRK, 2017

Table 16-24 summarizes the factored truck haulage cycle times and corresponding one-way haul distances from the pit for each year. These cycle times are the truck cycle times away from the loading unit, and do not include truck spotting and loading times.

Table 16-24: Factored Pit Haulage Cycle Times and One-Way Distances

	Saprolite				Waste Rock			
Period	Cycle Time min*		One Way Distances m		Cycle Time min*		One Way Distances m	
	Waste	Ore	Waste	Ore	Waste	Ore	Waste	Ore
Yr -2	18.1	16.2	2,441	1,983	15.7	13.5	2,073	1,650
Yr -1	21.7	17.1	2,627	2,024	20.6	13.5	2,769	1,650
Yr 1	20.4	17.2	2,626	2,257	18.8	17.0	2,614	2,294
Yr 2	16.1	12.9	2,187	1,508	13.3	12.0	1,787	1,650
Yr 3	13.3	11.7	1,683	1,595	13.7	11.3	1,698	1,476
Yr 4	15.9	8.9	2,584	1,004	17.0	10.4	2,731	1,198
Yr 5	11.5	12.5	1,654	1,891	16.6	14.0	2,328	1,880
Yr 6	14.5	10.3	2,323	1,485	15.7	11.9	2,564	1,749
Yr 7	12.7	18.1	1,989	2,997	18.7	17.0	2,793	2,316
Yr 8	13.1	18.4	1,835	2,988	12.4	15.0	1,601	2,259
Yr 9	12.7	17.0	1,794	2,708	14.7	18.3	1,891	2,770
Yr 10	N/A	N/A	N/A	N/A	19.8	21.5	2,837	3,311
Yr 11	N/A	N/A	N/A	N/A	25.3	21.9	3,635	3,421
Yr 12	N/A	N/A	N/A	N/A	N/A	N/A	N/A	N/A

Source: SRK, 2017
Note: *Truck cycle times away from loading unit including dumping time.
This does not include truck spotting and loading times.

Truck spotting and loading times were then added to the factored haul cycle times to make up total haul cycle times. Spotting and loading times used were taken from the loading unit time estimates. Loading time estimates used were averaged according to the use of the different loading units in the respective years.

Table 16-25 shows the pit waste haulage quantities of saprolite and waste rock for use in construction of the TSF embankments in particular years. These haulage costs were included in the mining costs. It was assumed that approximately 85% of the pit mined waste material could be directly placed in the TSF embankments by the mining trucks, and that 15% of the pit mined waste material would be placed nearby and would require re-handling by the TSF construction equipment.

Table 16-25: Haulage Quantities to TSF Embankments

Quantity Category	Material Type	Yr -2 kt	Yr -1 kt	Yr 1 kt	Yr 2 kt	Yr 3 kt	Yr 4 kt	Yr 5 kt	Yr 6 kt	Yr 7 kt
TSF Phase 1	Saprolite	-								
TSF Phase 1	Waste Rock	117								
TSF Phase 2	Saprolite		1,655	1,168	-					
TSF Phase 2	Waste Rock		282	7,692	1,829					
TSF Phase 3	Saprolite						3,201			
TSF Phase 3	Waste Rock						11,736			
TSF Phase 4	Saprolite								3,378	2,764
TSF Phase 4	Waste Rock								10,742	12,751
Total Saprolite	**Saprolite**	**-**	**1,655**	**1,168**	**-**	**-**	**3,201**	**-**	**3,378**	**2,764**
Total Hard Rock	**Waste Rock**	**117**	**282**	**7,692**	**1,829**	**-**	**11,736**	**-**	**10,742**	**12,751**

Source: SRK, 2017

With the dry material densities, loaded rock swell factors and moisture contents used, the truck capacities were limited by material weight (rather than truck body volume).

Table 16-26 shows selected hauling productivity information based on the planned hauling units in saprolite.

Table 16-26: Hauling Statistics by Unit Type in Saprolite

Hauling Equipment/ Material Type	Rated Truck Size t	Truck Fill Factor Dry – Wet Tonnage Basis %	Typical Total Truck Loading Time* by Excavator min	Total Truck Dumping Time min	Hauling Efficiency Factor %	Production per Unit (100% Available) t/op hr
ADT/Waste	40	77% - 100%	2.55	1.10	92% - 95%	77 – 133
ADT/Ore	40	77% - 99%	2.72	1.10	90% - 95%	87 – 159

Source: SRK, 2017
Note: * Includes truck spotting time.

Table 16-27 shows selected hauling productivity information based on the planned hauling units in waste rock.

Table 16-27: Hauling Statistics by Unit Type in Waste Rock

Hauling Equipment/ Material Type	Rated Truck Size t	Truck Fill Factor Dry – Wet Tonnage Basis %	Typical Total Truck Loading Time* by Excavator min	Total Truck Dumping Time min	Hauling Efficiency Factor %	Production per Unit (100% Available) t/op hr
ADT/Waste	40	95% - 100%	2.55	1.10	92% - 95%	82 – 153
ADT/Ore	40	95% - 100%	2.72	1.10	90% - 95%	93 – 175
Rear Dump/Waste	91	95% - 100%	3.34	1.20	92% - 95%	181 – 331
Rear Dump/Ore	91	95% - 100%	3.59	1.20	90% - 95%	202 – 370

Source: SRK, 2017
Note: * Includes truck spotting time.

Table 16-28 shows selected hauling productivity information based on the planned hauling equipment.

Table 16-28: Hauling Productivities by Unit Type

Loading and Hauling Equipment Types	Production per Unit (100% Available) t/op hr	Planned Operating Hours per Shift scheduled op hrs	Planned Operating Hours per Year * scheduled op hrs	Estimated Mechanical Availability ** %	Actual Operating Hours per Year op hrs	Annual Production Capacity per Unit Mt/y
ADT/Waste	82 – 153	10.2	7,242	85%	6,156	Variable
ADT/Ore	93 – 175	10.2	7,242	85%	6,156	Variable
Rear Dump/Waste	181 – 331	10.2	7,242	85%	6,156	Variable
Rear Dump/Ore	202 – 370	10.2	7,242	85%	6,156	Variable

Source: SRK, 2017
Note: * Includes allowance of 10 days downtime for weather delays.
** Typical mechanical availabilities for trucks used.

Truck hauling productivities were calculated for each type of truck for each year of the mining operations that were then used to estimate respective fleet hauling operating hours required, which were used as a basis for determining the truck fleet requirements.

In Year 4, haul truck availabilities were allowed to be increased to 88% as part of a specifically planned program to improve availabilities in that year, in order to avoid purchase of additional equipment units that would only be needed for that year.

The ADTs will be used for re-handling ore from the low-grade stockpile to the primary crusher. This will be a relatively short haul, approximately 830 m one-way. Additional hauling operations by the ADTs will include crushed waste backfill to be hauled to the pits for road and ramp surfacing.

16.4.6 Auxiliary Equipment

The crushing/screening plant will be used to provide crushed waste rock for site and mine road surfacing, and pad surfacing during pre-production/Project construction years, and thereafter mainly for mine road and ramp surfacing only.

The track dozers will be used for drill site preparation, road and ramp development, maintenance of loading areas, WRDs and stockpiles, and other duties. The wheel dozers will primarily perform general dozing and clean-up in areas not worked by the track dozers. The graders and water trucks will maintain ramps, haul roads, and operating surfaces. The vibratory compactors will be used in developing new roads or repairing existing roads. The (smaller) excavators will perform site development work including pioneering and drainage diversion ditch development. The major mining equipment fleet size for roads and dumps was based on the general production level, number of active working faces, and allowance for general site conditions (including annual precipitation).

Annual operating hours were estimated for all of the major mining support equipment units, in general, between 3,500 and 5,000 operating hours per unit per year were scheduled.

Mining support equipment includes equipment for equipment erection and repair (120 t crane), equipment movement around the site (low bed trailer for moving the drills and large excavators over longer distances around the mine site), which is pulled by a recovery truck. The recovery truck will be used to transport broken down mobile units back to the mine repair shops when required. Mining equipment maintenance units are included such as fuel trucks, which will deliver to tracked mining equipment in the field from the main fuel station, lube trucks, heavy duty mechanics' trucks, welders' trucks, tire service trucks, and a general purpose flatbed truck. Other mining support equipment includes personnel vans for taking operators to tracked units in the field, pickup trucks for mining staff, and light plants.

The mining department functions will include all geology, mine engineering and mine administration functions. The mine department will have mine surveying equipment, mine engineering and geology equipment (instruments, computers, peripheral computer equipment, software, etc.), and mine communications (radios) for the staff pickup trucks.

Grade control will be very important to ensure that the higher grade ore is mined with minimal dilution and is being sent directly to the crusher. Allowance has been made for assaying all necessary blasthole samples during the mining operations.

16.5 Mine Dewatering

Dewatering will be required for the open pit. A combination of run-on from areas up-gradient of the pit, precipitation falling within the outer perimeter of the pit and groundwater inflows into the pit will account for the total volume of water that will need to be handled by the dewatering equipment. SRK's pit perimeter boundaries and surrounding topography were used as the basis in determining the quantity of surface water to be pumped each year. Additional inflows from groundwater were estimated with a calibrated numerical model.

The groundwater flow model predicts maximum passive groundwater inflow into the open pit as high as 3,975 m^3/d (46 L/sec) during Year 10 of pit excavation. An average predicted pit inflow through LoM is 2,250 m^3/d (26 L/sec. Total maximum pit inflow, considering both net precipitation/surface run-off and groundwater flow, is predicted at 8,801 m^3/d (102 L/sec). The average total inflow to the pit following closure is predicted to be 5,668 m^3/d (65.6 L/sec). Approximately 40% of the predicted total inflow is coming from groundwater and the remaining water is sourced from direct precipitation and run-off. The primary sources of groundwater inflow to the pit are 1) captured groundwater in saprock that discharges to the pit from the south highwall, and 2) depletion of groundwater storage.

Most of the run-off generated by precipitation falling outside of the perimeter at the top of the pit will be diverted around the pit into various drainages adjacent to the pit. Precipitation inflow directly into the pit and pit groundwater inflow will be collected at the bottom of each pit phase in a series of sumps, pumped to the to the pit rim and from there channeled in accordance with the Site water Management Plan.

A site-wide water balance, developed for the Project was used to predict inflows, outflows and accumulation of water in the pit bottom. Pumps and piping for pit dewatering are included under Site Water Management.

16.5.1 Surface Water

The location of the pit on the north side of the Dékou-Dékou Massif results in areas of steep hillsides generating run-on to the pit. While extensive diversion ditches are proposed around the pit, the difficult terrain and marginal geotechnical stability of ditch construction in the saprolite makes it impractical to divert upgradient surface water around the pit. In addition to surface water run-on, tropical rainfall will produce heavy run-off from within the pit limits that will result in significant water reporting to the pit bottom. The water balance estimated surface water inflows to the pit would be in the range of 800 m^3/day during the wet seasons in the early mine life, but as the pit expands, this inflow is expected to increase to approximately 17,000 m^3/day during the wet months. These are average values and daily extremes may be several times larger. Run-off from within the pit limits is projected to increase steadily from an average of 1,000 m^3/day to 10,000 m^3/day by the end of mining.

16.5.2 Groundwater

Predictive simulations using a 3D numerical groundwater model were completed for passive inflow conditions, assuming no active dewatering with wells. The model predicted a maximum passive groundwater inflow up to 3,975 m^3/d (46 L/sec) during Year 10 of pit excavation. An average predicted pit inflow through LoM was 2,250 m^3/d (26 L/sec). Given the general low-permeability of the bedrock, SRK did not design an active dewatering system (series of dewatering wells around the pit).

16.5.3 Dewatering System

Pit sumps will be necessary in each lobe of the pit throughout the mine life. Pit lobes will form and join as the pit is developed, but in general, there are four separate pit lobes where excess water will accumulate. Under the mine dewatering plan, each pit lobe will include a sump and multiple 30 L/sec capacity diesel powered trash pumps capable of evacuating the sumps within a few days of heavy rainfall. However, water balance modeling predicts that periods of intense rainfall may result in excessive accumulation of water in the pit sumps for periods of weeks. During the majority of the mine life under most climatic conditions, peak water accumulation in the pit will be not exceed 0.5 Mm^3 during the peak of the wet season, distributed amongst multiple pit sumps. The model predicts that the sumps can be completely evacuated at the end of every wet season until the last year of mining, when the west lobe of the pit is inactive and can be used for water storage if excessive water accumulation is experienced.

Pit sumps pumps have been selected for flexibility and portability, so that they can be relocated as necessary to address the changing pit geometry and work around mining activities. The mine dewatering plan includes a central booster station, located within the pit at approximately mid-depth, to reduce the lift required from the portable pit sump pumps. The booster station will lift the pit sump water the remaining distance to the pit rim, where it will flow by gravity to the CWP, located between the CWRD and the TSF. Water from the pit sumps will mix with other contact water from the Project for consumption in the milling processes, used as dust control, or treated and discharged as necessary to maintain adequate surge capacity in the contact water system.

The water balance model indicated a combined pumping capacity of approximately 360 L/sec, not including standby pumps, was needed by the end of the mine life to limit the accumulation of water in the pit sumps. This pumping capacity will steadily increase over the mine life and will be distributed around the different pit sumps as needed based on pit geometry and localized groundwater inflows. Over the LoM, the water balance predicted that average annual pumping flows will range from 0.2 Mm^3 (average of 550 Mm^3/day) during pre-production to 4.8 Mm^3 (average of 13,150 Mm^3/day) during the last year of mining. The in-pit booster station is not anticipated until Year 5, when the pit deepens significantly.

Upon the cessation of mining activities, the pit will be allowed to flood. Rapid filling of the pit at the end of mining is desirable from a geochemical and water quality standpoint. Diversion ditches upgradient of the pit will be reclaimed and/or converted to safety berms, and treated contact and process water will be directed to the forming pit lake, instead of discharged below the MSA. The water balance model predicts that the pit will fill within six years. Once the pit Lake reaches the pit rim, the pit will spill to the West, discharging into Violette Creek, tributary to the Roche River. Geochemical modeling has predicted that water quality in the pit Lake by this time will be acceptable for discharge to the environment (SRK, 2017b).

17 Recovery Methods

17.1 Process Selection and Flowsheet

The process plant design, derived from the interpretation of the test work results, reflects a robust metallurgical flowsheet designed for optimum recovery with minimum operating costs and utilising unit operations that are well proven in industry. The key criteria for equipment selection are suitability for duty, reliability and ease of maintenance. The plant layout provides ease of access to all equipment for operating and maintenance requirements whilst maintaining a compact footprint that will minimize construction costs.

The key Project and ore specific criteria for the plant design are:

- 4.6 Mt/y (12,330 t/d) throughput based on the design ore blend of 89% felsic tuff, 7% granodiorite and 4% mafic;
- Mechanical availability of 91.3% supported by crushed ore storage and standby equipment in critical areas; and
- Sufficient instrumentation and automation to achieve design production rates, to enable stable process operations and to facilitate safe operation.

The Montagne d'Or plant has been designed to treat the range of ore types and blends that will be mined over the life of the Project.

17.1.1 Selected Process Flowsheet

The treatment plant design incorporates the following unit process operations:

- Primary jaw crushing to produce a coarse crushed product;
- A crushed ore surge bin with bin overflow conveyed to a dead stockpile. Ore will be reclaimed from the dead stockpile to feed the milling circuit when the crushing circuit is off-line;
- A single stage semi-autogenous grinding circuit with recycle crushing (SS SAC) circuit with a 14 MW SAG mill in closed circuit with a pebble crusher and hydrocyclones to produce an 80% passing 75 micron grind size;
- Gravity concentration and removal of coarse gold from the milling circuit recirculating load and treatment of gravity concentrate by intensive cyanidation and electrowinning to recover gold to doré;
- Pre-leach thickening to increase the slurry density feeding the (CIL) circuit to minimize CIL tankage, smooth out fluctuations in the milling circuit, improve slurry mixing characteristics and reduce overall reagent consumption;
- Leach/CIL circuit incorporating a leach tank and six CIL tanks with carbon for gold adsorption providing a total of 31 hours leach time;
- A 10 tonne split AARL (Anglo American Research Laboratories) elution circuit treating loaded carbon, electrowinning and gold smelting to produce doré;
- Tails wash thickener to reduce the weak acid dissociable cyanide (CN_{WAD}) contained in the CIL tails prior to cyanide destruction and to recover free cyanide (CN_{FREE}) in the process water and recycle to the milling circuit;

- An SO_2/Air cyanide destruction circuit to reduce the tailings CN_{WAD} concentration to below 10 ppm; and
- Tailings pumping to the TSF.

A simplified flowsheet for the Montagne d'Or process plant is shown in Figure 17-1.

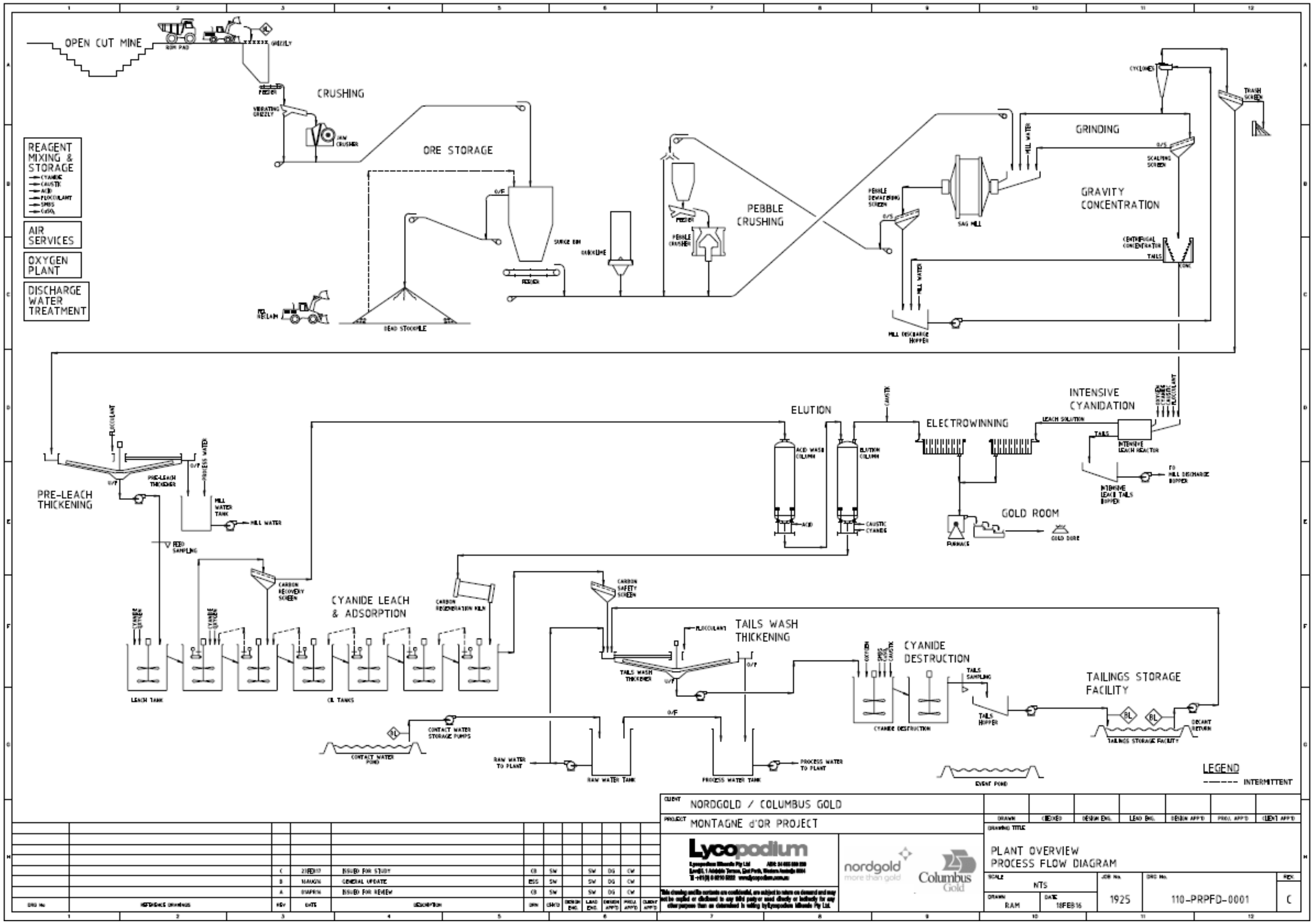


Figure 17-1: Montagne d'Or Plant Simplified Flowsheet

17.2 Key Process Design Criteria

The key process design criteria listed in Table 17-1 form the basis of the detailed process design criteria and mechanical equipment list.

Table 17-1: Summary of Key Process Design Criteria

	Units	Design Criteria	Source*
LoM Ore Blend		76% Felsic / 4% Granodiorite / 11% Mafic / 5% Saprolite / 3% Saprock	Client
Design Blend		89% Felsic / 7% Granodiorite / 4% Mafic	Client
Plant Capacity	t/y	4,500,000	Client
Head Grade	g Au/t	2.0	Client
Head Grade	g Ag/t	4.0	Client
Head Grade	%Cu	0.1	Client
Gold Extraction*	%	97	Test work
Silver Extraction*	%	60	Test work
Copper Extraction*	%	6	Test work
Crushing Plant Utilisation	%	85.0	Lycopodium
Plant Utilisation	%	91.3	Lycopodium
Crushing Work Index (Cwi)	kWh/t	16.6	OMC
SMC A x b		36.0	OMC
Bond Ball Mill Work Index (Bwi)	kWh/t	11.9	OMC
Abrasion Index (Ai)		0.06	OMC
Grind Size P_{80}	µm	75	Test work
Grinding Media Consumption	kg/t	0.443	OMC
Gravity Gold Recovery	%	40	Test work
Gravity Silver Recovery	%	20	Test work
Pre-leach Thickener Solids Loading	$t/m^2/h$	1.0	Test work
Required Leach Circuit Residence Time	hrs	24	Test work
Selected Leach Circuit Residence Time	hrs	31	Client
Leach Slurry Density	% w/w	51	Lycopodium
Number of Leach Tanks		1	Test work
Number of Adsorption Tanks		6	Lycopodium
Leach Cyanide Consumption^	kg/t ore	0.55	Test work
pH Modifier		Quicklime	Lycopodium
Leach Lime Consumption (90% avail. CaO)^	kg/t ore	0.61	Test work
Elution Circuit Type		Split AARL	Lycopodium
Elution Circuit Size	t	10	Lycopodium
Frequency of Elution	strips/week	6	Lycopodium
Cyanide Destruction Method		Air/SO_2	Lycopodium
Tailings CN_{WAD} Concentration	ppm	<10	Client
Excess Contact Water Treatment	Peak L/sec	180	SRK
Excess TSF Water Treatment	Peak L/sec	140	SRK

Source* 'Client' refers to advice from Nordgold/Columbus.
'Lycopodium' refers to Lycopodium experience or generally accepted practice.
'Test work' refers to PEA and BFS test work conducted at BV Vancouver.
'OMC' refers to advice from Orway Mineral Consultants.
'SRK' refers to advice from SRK.
Extractions at design head grades.
Reagent consumptions allow for losses to tails.

17.3 Process and Plant Description

17.3.1 RoM Pad

The RoM pad will be used to provide a buffer between the mine and the plant. The RoM stockpile will allow blending of ore feed stocks, and will ensure a consistent feed type and feed rate to the plant.

RoM ore will be loaded into the crushing circuit feed bin (RoM bin) by direct tipping from the mining haul trucks or by FEL.

17.3.2 Crushing and Grinding Circuit

RoM ore will be drawn from the RoM bin at a controlled rate by an apron feeder and discharged onto a vibrating grizzly. The grizzly oversize will report to the jaw crusher for primary crushing. The jaw crusher product together with grizzly undersize will report to the crusher discharge conveyor feeding directly to the crushed ore surge bin.

Ore will be withdrawn from the surge bin and fed via the mill feed conveyor to the SAG mill. Surge bin overflow will report to the conveyor feeding a dead stockpile. Ore from the dead stockpile will be loaded by FEL back into the surge bin to maintain mill feed when the crushing circuit is off line. Lime and SAG mill grinding media will be added to the mill feed conveyor as required.

The grinding circuit will consist of a SAG mill in closed circuit with a pebble crusher and hydrocyclones. Crushed ore will be fed directly to the SAG mill with process water added to the SAG mill to achieve the required milling density.

The SAG mill will discharge via a trommel onto the pebble dewatering screen. Screen oversize, consisting of pebbles and worn steel grinding media, will discharge onto the pebble discharge and pebble transfer conveyors. Worn media will be removed by a magnet and pebbles will report to the pebble crusher with crushed pebbles conveyed back to the mill feed conveyor. The facility to by-pass the crusher and recycle pebbles directly to the mill feed conveyor will be provided and is expected to be used for the majority of ore feed blends.

Trommel and screen undersize will gravitate to the mill discharge hopper and will be pumped to the hydrocyclone (cyclone) cluster for size classification.

The cyclone underflow (coarse material) will gravitate to the SAG mill feed chute for further processing. A portion of the cyclone underflow will feed the gravity circuit. The cyclone overflow (product size material) will gravitate to the trash screen for removal of trash material and coarse particles. Trash screen underflow will report to the pre leach thickener.

17.3.3 Gravity Circuit

Feed to the gravity circuit will be split between two parallel gravity trains, each consisting of a scalping screen and a centrifugal concentrator. The scalping screens will remove coarse material which will gravitate to the SAG mill feed hopper for further processing. The screened slurry will be processed in automated centre discharge centrifugal concentrators to recover free gold and silver to the gravity concentrate.

The gravity concentrate will report to the goldroom for further processing and the gravity tails slurry will gravitate to the mill discharge hopper for further processing.

17.3.4 Pre-leach Thickening

Trash screen underflow from the grinding circuit will be thickened in a high rate thickener. The thickener feed slurry will be mixed with flocculant and will report to the pre-leach thickener.

The thickened slurry (thickener underflow) will be pumped to the leach feed distribution box and the pre-leach thickener overflow will gravitate to the adjacent mill water tank and will be distributed as dilution water to the milling circuit. Process water from the tails thickener overflow tank and the process water tank will be added to the mill water tank to balance mill water requirements.

17.3.5 Leach and Carbon Adsorption Circuit

The adsorption circuit will consist of one leach tank and six carbon-in-leach (CIL) adsorption tanks providing 31 hours residence time. The tanks will be interconnected with launders and slurry will flow by gravity through the tank train.

Pre-leach thickener underflow will be pumped to the CIL circuit. Quicklime added to the mill feed conveyor will ensure that the slurry pH is suitable for cyanidation and sodium cyanide solution will be metered into the CIL circuit. Oxygen will be sparged into the tanks to provide oxygen to the leach slurry. A high shear mixer will be used to contact oxygen with the slurry in the leach tank in order to increase the slurry dissolved oxygen level quickly and efficiently. Oxygen will also be sparged down the shafts of the CIL agitators.

Barren activated carbon will be added to the last CIL tank and advanced counter current to the slurry flow. The leached gold will adsorb onto the carbon and be removed from the CIL slurry. Carbon loaded with gold (loaded carbon) will be recovered from the slurry from CIL Tank 1 via the loaded carbon recovery screen and will report to the elution circuit.

Slurry from the last CIL tank (CIL tails) will gravitate via the carbon safety screen to the cyanide destruction circuit.

As the Montagne d’Or ores contain copper of which a small portion is cyanide soluble, the CIL circuit will be operated at a free cyanide level and pH that will minimize copper adsorption onto the carbon.

17.3.6 Elution and Gold Recovery

The following operations will be carried out in the elution and goldroom areas:

- Acid washing of loaded carbon;
- Cold cyanide wash of loaded carbon;
- Stripping (elution) of gold and silver from loaded carbon using the split AARL method;
- Electrowinning of gold and silver from pregnant solution;
- Smelting of electrowinning products;
- Regeneration of barren carbon; and
- In-Line Leach Reactor (ILR) and dedicated electrowinning cell for treatment of gravity concentrate.

Loaded carbon will be washed with a dilute acid solution to remove contaminants prior to being rinsed with water. Cold cyanide washing to remove the majority of copper from the loaded carbon will then be completed followed by a water rinse. The loaded carbon will be eluted with a hot dilute cyanide/caustic solution which will recover the gold and silver from the carbon into the solution. The gold/silver solution (pregnant solution) will be pumped through electrowinning cells and the gold and silver will be recovered onto the cell cathodes. The gold and silver will be removed from the cathodes by high pressure water jets with the gold/silver sludge being filtered and dried prior to smelting with fluxes in a furnace to produce doré bars.

Eluted carbon (barren carbon) will be transferred to the carbon regeneration kiln for reactivation prior to re-use in the CIL circuit.

Gravity concentrate will be leached at high levels of cyanide and oxygen in an in-line leach reactor to extract the contained gold and silver into solution. The solution will be pumped through the dedicated gravity electrowinning cell with the gold and silver sludge recovered from the cathodes and smelted. Gravity tails will be pumped to the mill discharge hopper for additional processing.

Fume extraction equipment will be provided to remove gases from the electrowinning cells, oven and smelting furnace.

17.3.7 Tails Wash Thickening

Underflow from the CIL circuit carbon safety screen will gravitate to the cyanide destruction circuit. Low cyanide decant return water and raw water make up will be used to dilute the CIL tails prior to thickening. Flocculant will be added to the diluted slurry which will report to the tails wash thickener.

Tail wash thickener overflow will gravitate to the thickener overflow tank and will be pumped to the mill water and process water tanks for re-use in the process. The washed thickener underflow slurry will be pumped to the cyanide destruction circuit.

17.3.8 Cyanide Destruction

Cyanide destruction will be carried out using the air/SO_2 process which will reduce the CN_{WAD} in the slurry to less than 10 mg/L prior to discharge from the plant. The cyanide destruction circuit will consist of two tanks providing one hour residence time. The tanks will be interconnected with launders to allow the circuit to be run in parallel or series.

Underflow from the tails wash thickener will be pumped to the cyanide destruction circuit. Copper sulphate and sodium metabisulphite (SMBS) solutions will be added to provide the required copper and sulphur dioxide for the cyanide destruction process. Oxygen from the oxygen generation plant will be sparged down the shafts of the cyanide destruction agitators to provide oxygen to the slurry. Provision will be made for caustic solution to be added to maintain a slurry pH 8.0 to 9.0.

Treated tailings will gravitate to the tailings hopper and will be pumped to the TSF.

17.3.9 Tailings Disposal and Decant Return

Tailings will be deposited into the TSF using established discharge and decant methods.

Supernatant water (decant return) will be recovered from the TSF and returned as low cyanide process water to the plant for re-use. The majority of the decant return will be added to the process as wash water for the CIL tails. Excess TSF decant return water not required for processing will be discharged from the site following treatment.

17.3.10 Reagents& Consumables

Sufficient stocks of reagents and consumables will be stored on site to ensure that supply interruptions due to port, transport or weather delays do not restrict production. The following reagents and consumables will be used in the process:

- Grinding media (steel grinding balls);

- Quicklime;
- Sodium cyanide (briquettes);
- Caustic soda;
- Hydrochloric acid;
- Activated carbon;
- Flocculant;
- SMBS;
- Copper sulphate; and
- Smelting fluxes.

17.3.11 Services

The following plant services will be provided:

- Raw and process water supply including recycle/reuse of tailings decant water and contact water from pit dewatering and dump run-off;
- Fire water;
- Potable water;
- Plant and instrument air; and
- Oxygen for the leach circuit.

17.3.12 Excess Site Water Treatment

Excess site water will be treated in WTPs to produce water that will meet the environmental requirements for discharge to the environment.

It is anticipated that excess contact water from the open pit and WRD run-off will require treatment from early in the Project life.

With a net positive water balance, it is estimated that the site will eventually generate an excess of TSF decant return water by Year 3. This will also be treated before discharge.

17.4 Plant Layout and Design Considerations

The civil, mechanical and electrical design of the plant facilities is based on standards deemed appropriate for the local climatic conditions and reflective of the local topography. The plant layout and equipment selection is attentive of the requirements necessary for Project implementation.

17.4.1 Site Selection

The plant site geotechnical investigation conducted by SRK has confirmed that ground conditions will require improvement. The appropriate site preparation, excavation, fill and drainage has been adopted in the design. Prior to construction of the plant, test pits will be located at the appropriate locations to enable an engineered system to be developed specific to the plant area.

The process plant site layout is shown in Figure 17-2. The plant has been located on the edge of and above the floor of the valley running south from Camp Citron. The plant site is located below two sediment ponds used to control discharges from the pit and haul road areas. Outlets from the ponds have been sized to accommodate the 100-yr flood without impacting the plant area. The water discharge will be routed west around the plant site to return to the original drainage. A final flood plain analysis will be performed based on the final grading of the drainage ways adjacent to the plant site prior to construction.

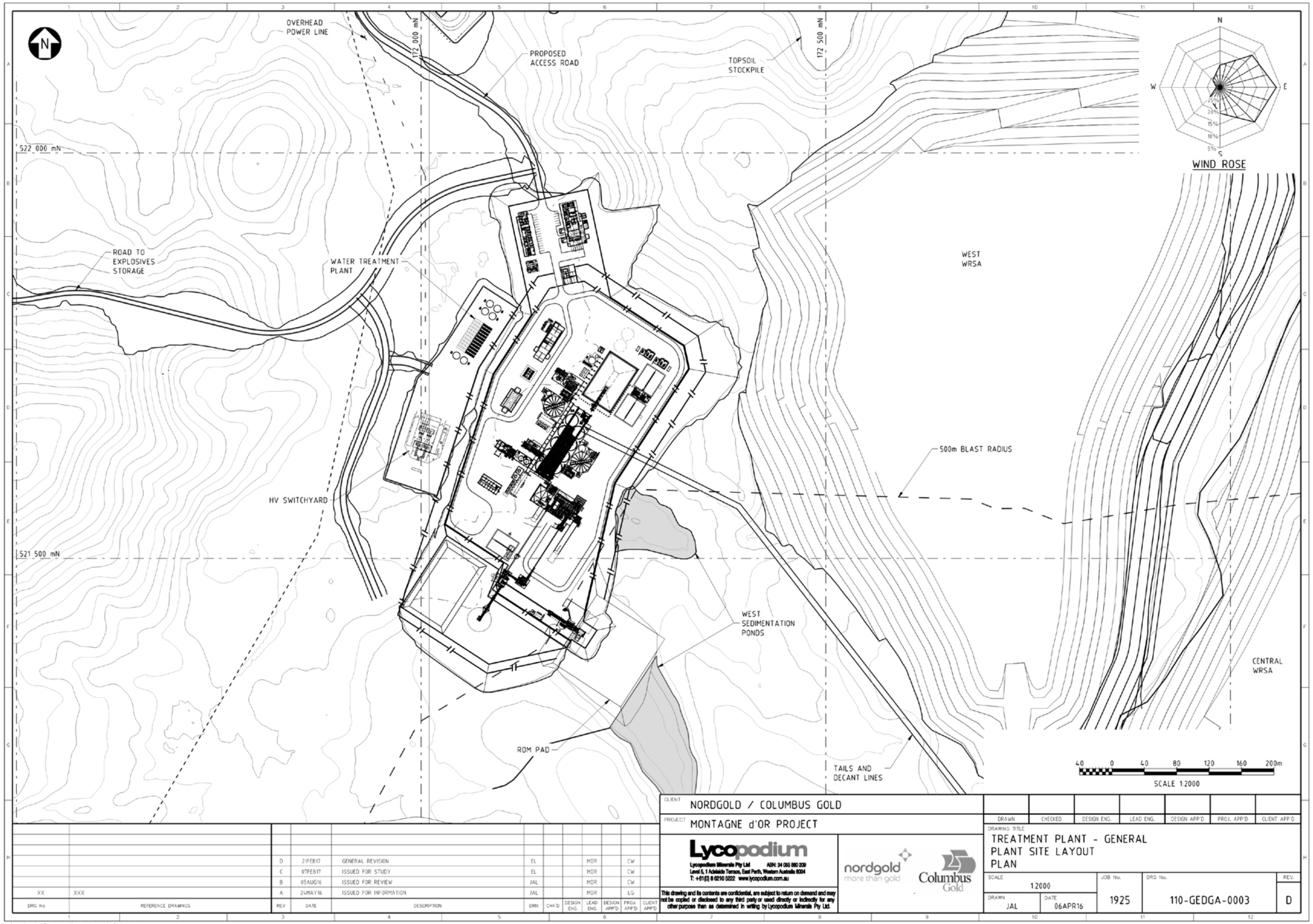


Figure 17-2: Montagne d'Or Process Plant Site Layout

17.4.2 General Design Considerations

Process equipment has been selected to meet the process duty point derived from the process design criteria. A suitable design factor has been added to the duty requirements to account for the minor process interruptions and the inevitable variability that occurs in an operating plant.

Typically manufacturers' standard equipment to EU compliance requirements has been adopted for design and costing purposes. The equipment has been selected on the basis of its criticality to the process and suitability for use within the operating environment.

For the management of cyanide, the design addresses reagent unloading, storage, handling, containment and detoxification of cyanide containing process streams. The cyanide handling area has been located remote from offices and workshops and packaging will be disposed of by incineration. This design approach aligns with the requirements of the International Cyanide Management Code as well as local regulatory requirements for dangerous goods.

The design also addresses the environmental requirements for the Project. This includes the provision of suitable dust suppression, fume extraction and treatment, discharge stream treatment and bunding of relevant areas.

17.4.3 Geotechnical Foundation Design Considerations

The plant site field program consisted of one diamond drillhole to a depth of 38 m and two test pits to a depth of 3.5 m. Shelby type samples were collected at 1.5 m intervals. Saprolite soils were measured to a depth of 30 to 40 m. Saprock and bedrock units are below the saprolite. The saprolite is intensely weathered to decomposed and is more adequately described using engineering soils terminology. Soils were logged using the unified soil classification system (USCS). Some relict textural and structural features of the parent rock may be observed. Intact field strengths as measured by a pocket penetrometer of less than 500 kiloPascals (kPa) are typical. Undrained shear strength estimates of less than 100 kPa were measured. Void ratios of 0.8 to 1.1 were typical in the soil. These typically occur at a depth range of 2 to 30 m.

The results of the laboratory test program indicates that moisture contents of the saprolite soil samples, as tested in the current program, are approaching and are within 2% of the liquid limit (LL) and plastic limit (PL) of the saprolite soil. When the soils reach these limits they have the potential to flow and creep. As a result, the material is unsuitable for use as an engineered fill and would need to be blended with another material to bring into an engineered fill gradation specification. This presents a problem for bearing capacity of large foundation loads. If the water contents were increased by 2% to 3% the saprolites will approach the LL at which point the material would begin to flow and collapse due to the high void ratios of the saprolite. SRK recommends using the undrained strengths for design.

Foundation design criteria are summarized in Table 17-2, giving building load minimum requirement of footing embedment depths and expected settlements for each building foundation element based on allowable bearing capacities. Excessive settlements in the range of 20 to 30 cm would be expected for conventional mat and spread footing foundations.

Table 17-2: Plant Site Structure Building Loads, Elastic Settlement, Consolidation Settlement, Minimum Embedment and Allowable Loads for Drained and Undrained Conditions

Structure	Approx. Foundation Size (m x m)	Footing Type	Applied Working Load (kPa)	Elastic Settlement			Consolidation Settlement (OC) (cm)	Min. Embed. (m)	Drained			Undrained			
				Edge (cm)	Centre (cm)	Delta (cm)			q_all (global) (kPa)	q_ult (global) (kPa)	FS Drained	q_all (global) (kPa)	q_ult (global) (kPa)	FS Undrained	Ok?
# Primary Crushing	140 x 140	Static Stiff	400	6.2	6.2	0.0	14.6	18.9	5359	16078	40.2	401	1202	3.00	ok
# Stockpile	50 x 50	Static Flexible	400	2.8	7.3	4.6	14.6	18.9	3556	10667	26.7	401	1202	3.00	ok
## Leach Feed Thickener	20 dia	Static Flexible	80 avg, 300 peak	6.4	17.4	11.0	24.0	3.0	1454	4362	14.5	300	901	3.00	ok
## Tailing Thickener	20 dia	Static Flexible	80 avg, 300 peak	6.4	17.4	11.0	24.0	3.0	1454	4362	14.5	300	901	3.00	ok
## CIL Tanks	10	Static Flexible	150 avg, 300 peak	4.6	11.0	6.3	24.0	3.0	1102	3305	11.0	300	901	3.00	ok
Miscellaneous	1 x 1	Static Flexible	300	0.7	1.5	0.8	24.0	3.0	788	2365	7.9	300	901	3.00	ok
Pad Footings	2 x 2		300	1.4	2.9	1.5	24.0	3.0	828	2485	8.3	300	901	3.00	ok
	4 x 4		300	2.6	5.6	3.0	24.0	3.0	908	2724	9.1	300	901	3.00	ok
Miscellaneous Raft	10 x 10	Static Flexible	300	5.0	12.0	7.1	24.0	3.0	1147	3441	11	300	901	3.00	ok
Miscellaneous Strip Footings	0.75	Static Flexible	300	0.5	1.1	0.6	18.3	13.3	1772	5317	17.7	300	900	3.00	ok
	1		300	0.7	1.5	0.8	18.3	13.3	1785	5355	17.8	300	900	3.00	ok
	1.5		300	1.0	2.2	1.1	18.3	13.3	1810	5429	18.1	300	900	3.00	ok

[1] These footings are located beneath stockpile or RoM pads. Size is the estimate size of earthworks. Applied load stated is for concrete
[2] Expectation for these facilities is soil stress fields will interact. Average applied loads equate to total vertical load divided by total loaded area of the facility. Peak loads reflect actual loads on discrete footings

Because of the observed weak and potentially collapsible saprolite conditions, driven piles are the recommended foundation solution for structures that are sensitive to differential settlements (e.g., the primary crushing, Stockpile facilities and other Miscellaneous and stripped footings). The driven piles should be battered in symmetric pile groups (batter range 5° to 10°) for example 3 x 3 or 9 x 9 pile groups. The battering of the piles provides lateral stability for the foundation. An engineered fill cap, 1 m thick, should be constructed to the engineered fill specification, by either adding waste rock, and/or by lime treating the saprolitic soils to meet the engineered fill specification.

SRK is of the opinion that the data at this level of study is sufficient for feasibility-level design and costing to +15%/-10%. Additional field investigations are required prior to final design. Drilling with a SPT or CPT is required to assess foundation soils conditions for final design. The additional field work should consist of 20 to 30 additional holes with SPT logging and/or CPT data located along the foundation alignments. The number of drillholes may be reduced if there is a geophysical survey of the saprolite, saprock and bedrock contacts. Additionally, a geophysical investigation should be conducted to determine the depth of bedrock contour and determine dynamic soil constants. Additional geotechnical characterization, laboratory and field testing of saprolite soils, and planned additives including waste rock and lime need to be conducted to provide data to bring cost estimates to a final design level.

17.4.4 Plant Layout

The plant equipment has been arranged to satisfy the following criteria:

- Grouping of process equipment either by circuit or similar equipment type, to facilitate containment and recovery of spilt material and to assist control and operability;
- Ease of access for maintenance;
- Use of gravity, where possible, to minimize transfer equipment, especially for solid materials; and
- Logical flow of material from one end of the plant to the other whilst being cognisant of process flow requirements.

The plant and infrastructure layout and location was developed with consideration given to site access, proximity to the pit, waste storage areas and optimisation of site preparation effort and is presented in Figure 17-3.

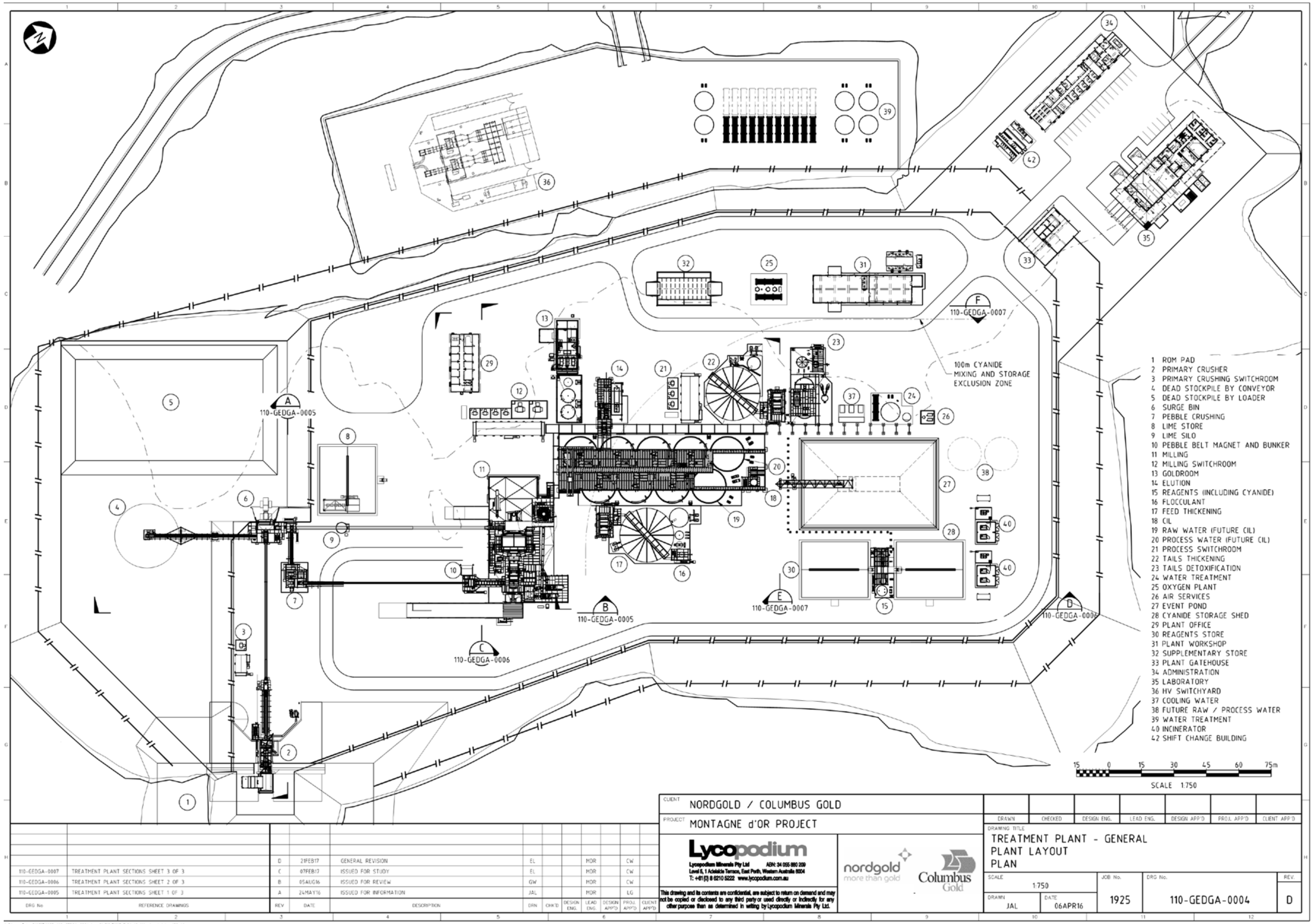


Figure 17-3: Montagne d'Or Process Plant Layout

17.4.5 Electrical Design

Electrical power will be provided to the site from the grid via a purpose built 106 km, 90 kV powerline. The point of connection to the Electricité de France – Systèmes Energétiques Insulaire (EDF-SEI) national grid will be the existing Margot HV substation located outside SLM which will require expansion of an additional switchyard bay. A step-down substation will be located adjacent to the plant site to distribute grid power within the site.

The installed load and maximum demand for the site is shown in Table 17-3. The maximum demand is calculated for a ½ hour window and represents the minimum supply capacity required for the site.

Table 17-3: Site Power Demand

Area	Connected Load	Estimated Maximum Demand	Estimated Average Continuous Load
Process Plant	29 MW	21.5 MW	18.5 MW
Infrastructure/Camp	2 MW	1.5 MW	1.0 MW
Totals	**31 MW**	**23 MW**	**19.5 MW**

18 Project Infrastructure

The Project is approximately 120 km by road from the nearest settlement (SLM) with good transport links, communications services and with a connection to the regional power grid. Existing infrastructure at site is limited and designed to support an exploration camp only.

18.1 Off-Site Infrastructure and Logistics Requirements

18.1.1 Port Infrastructure and Off-site Logistics

Project materials and procured equipment will be delivered to the Project site via the ports of Cayenne or SLM. Both port facilities have limited materials handling capacity and no heavy lift capacity. Accordingly, it will be necessary to deliver the materials and equipment in self-unloading vessels and unload directly on to road transport located alongside the vessel. Open laydown areas are available close to the Cayenne port.

Alternatively, the port of Paramaribo in neighbouring Suriname could be used but this would require a ferry crossing at SLM. It may also require a transfer from Suriname based trucks to those belonging to local transport providers.

Barges are available in the region and trans-shipment from Paramaribo and Cayenne by barge to SLM is feasible.

Based upon an assessment of road conditions between Cayenne and SLM, the port of Cayenne will be restricted to receiving standard gauge materials (i.e. 6.0 m and 12.0 m containers) with a payload maximum of 28 tonne. The existing transport corridor from Cayenne to SLM has a number of weight restricted bridges and within a section of the road between Laussat and SLM there are dimensional restrictions as a consequence of a truss bridge.

In light of the road transport constraints between Cayenne and SLM, the port of SLM is the best option for sea freight unloading and transporting directly to the Project site.

Consideration will be given to the storage and transportation of dangerous goods. There is no designated dangerous goods area available within the area of Cayenne or SLM to store materials such as reagents (cyanide). Should the Project be unable to obtain permits for short term storage at the port, arrangements will be made for offloading direct onto trucks for immediate transport to the site storage facilities. Adequate storage facilities at site have been incorporated into the Project scope.

18.1.2 Site Access

The Project is accessible via a 120 km seasonal forest road from the town of SLM, where the port of St. Laurent is located, or by helicopter/light aircraft to the Project's base camp at Camp Citron.

The current condition of the public section of the road between SLM and Apatou Crossing road is fair to poor and will need repair and maintenance during the Project construction and on-going operation phase. The general road condition and increase in construction/mining traffic may accelerate deterioration of the road and give rise to a heightened potential of incidents with local traffic. An upgrade of the access road from SLM to Apatou Crossing is to be considered.

The access road from Apatou Crossing to Citron is a private road although it is used by multiple organisations. It is in very poor condition and impassable to vehicles without 4WD and even light 4WD vehicles have difficulty negotiating the road during the worst of the wet season. The scope of works includes the upgrade of the existing main access road from Apatou Crossing to Citron, a distance of approximately 54 km.

Site Access Road Upgrade

The 53.5 km site access road between Apatou Crossing and Camp Citron is a private road but, as it is the only access to the area, it is used to access areas other than Camp Citron/Montagne d'Or.

The terrain between Apatou Crossing and Camp Citron is very rugged and the existing unsealed road crosses a number of water courses. It would appear from the alignment of the road that it was constructed by following contours during construction resulting in many twists and turns but with minimal requirements for cut and fill to straighten the alignment.

The track will require significant upgrading in order to accommodate the additional traffic generated by construction and operation of the Project. The upgraded access road will follow essentially the alignment of the existing road, however, sections will be improved to ensure that gradients and radius of bends are suitable for standard road tractor/trailer combinations and that a minimum design speed of 20 km/h is achieved for the entire length of road to keep transit times to a reasonable duration.

Bridges

Eleven existing bridges on the site access road have been identified as requiring repair or replacement. Allowance has been made in the road upgrade estimate to replace the bridges with new structures using a combination of single lane bridges constructed from local timber, which is a common form of construction in French Guiana, or prefabricated steel or concrete units.

18.1.3 Power Supply

The French Guiana grid is not connected to the grids of any neighbouring countries.

A 106 km, 90 kV overhead powerline will connect the site to the existing grid. The point of connection is at the Margot HV substation located outside SLM. It is anticipated that by the time the Project is constructed the power authority will have constructed a new regional power station in the vicinity of SLM to supplement the current mix of hydro and fossil fuel generated power.

In order to minimize the environmental impact of the powerline on the forest the powerline will be constructed as close as possible to the existing road to Apatou Crossing and the upgraded road to Camp Citron. Although some additional clearing will be required to broaden the road easement to accommodate the powerline this will also have a beneficial impact on the road as the additional sunlight will facilitate drying of the road surface after rain and reduce traffic damage. The cost of the additional powerline towers to accommodate the frequent changes in direction to stay adjacent to the road have been allowed for in the costs.

A step-down substation, comprizing two 30 MVA 90/11 kV ONAF transformers with auto-tap changers will be located at the mine to distribute within the site. The transformers are rated for the required site duty with N+1 redundancy.

18.2 On-Site Infrastructure

18.2.1 Site Accommodation

Pioneer accommodation will be provided in Camp Citron (44 bed capacity), supplemented by additional portacabin type rooms as required.

Construction accommodation will rely on the early completion of the 482-room permanent accommodation camp supplemented by Camp Citron, and portacabin type temporary units. It is estimated that construction numbers will peak at about 900 people requiring two beds to a room for part of the construction duration.

During operations, accommodation will be provided on site on a single status bus in/bus out basis from SLM on a two weeks on/two weeks off roster. Total employment at site is estimated to be 658 with an average of 285 on site at any one time. Allowance has been made for additional beds for visitors, contractors, short term maintenance personnel etc.

The accommodation camp potable water facility and sewage treatment plant will service the whole Project site.

18.2.2 Other Site Facilities

Other site facilities include:

- A MSA with mine office, changerooms, heavy and light vehicle workshops, warehouse and vehicle washdown facility;
- An explosives facility and magazine;
- General administration facilities including offices, clinic, training centre and assay and environmental laboratory;
- Process plant support facilities including a plant security gatehouse, changerooms, offices, workshop and warehouse and reagent storage sheds;
- Fuel storage and dispensing facility with a capacity of 1.25 ML of diesel (two weeks supply);
- Communications network including external telephone and data connections, LAN and WiFi network, private mobile radio system for operations and emergency use, mobile phone coverage, CCTV/security/access control system and camp entertainment system;
- WTPs to treat surplus site water contaminated by contact with process streams and/or potential acid mine drainage; and
- Waste disposal facilities including domestic waste incineration, waste oil and lubricant storage, collection, sorting and recycle of wastes and scrap.

18.2.3 Site Water Management

Surface water controls will be necessary around the Project area to limit erosion, control run-on to areas of active mining or waste placement, and to limit the amount of contact water that must be addressed by the mine water management system.

Under the surface water management plan, diversion ditches have been designed upgradient of all major facilities where topography and soil stability conditions allow. Approximately 15 km of surface water diversion ditches, roadside channels, downchutes, and spillways have been designed to control surface water run-on to Project facilities and infrastructure. Approximately 1/3rd of the surface water infrastructure is scheduled for construction during the pre-production/first year of mining.

Additional channels and ditches are scheduled as the Project facilities expand through time. Several of the diversions constructed early in the mine life will be covered by the expanding WRDs and will be replaced with new diversions above the expanding facility.

In general, diversions have been sized to convey the peak flow from the 100-yr, 24-hr storm event. The exception to this is the TSF closure spillway, design to convey diverted run-on flow during operations, and closure cover run-off from the Probable Maximum Flood (PMF) resulting from the Probable Maximum Precipitation (PMP) should that event occur during closure. The spillway has also been sized to convey the PMF discharge from the TSF if the PMP is experienced during the last phase of the TSF. During the previous 3 TSF phases, the PMP event will be contained within the TSF impoundment.

Diversions around the TSF have been designed to convey the 100-yr, 24-hr peak flow, but due to size constraints on the perimeter berm of the TSF, flows above the 10-yr, 24-hr peak flow will be allowed to discharge into the TSF, while the remainder of the flow will still be conveyed in the diversion. At storms less than the 10-yr, 24-hr event, all diverted flow will be conveyed around the TSF. This significantly reduces the peak flow in the diversions during the 100-yr, 24-hr storm event, but results in less than 5% of the run-off on an annual basis entering the TSF instead of being diverted around the facility.

All non-contact flows diverted around the mine facilities will be discharged into natural drainages around the site. However, all discharges will be routed through at least one sediment control pond before being released off-site. The sediment control ponds will utilize active sediment control techniques such as silt curtains to control sediment discharges during construction periods when sediment loading is high. Once natural vegetation has re-established on the disturbed hillsides, the sediment control ponds will be used as detention ponds to reduce sediment loading to the natural detain storm flows.

Run-off from the active mine facilities, such as WRDs, stockpiles and open pits, will be collected through internal ditches and collection sumps before being routed to the CWP. In general, the collection sumps cannot flow by gravity to the CWP and the water will be pumped to the CWP. The ponds are designed to contain all run-off and seepage from the facilities for storm events up to the 100-yr, 24-yr rainfall. Water that comes in contact with active mining facilities will not be released to the environment unless it meets applicable surface water quality standards.

The milling process requires between 10 to 60 L/sec of contact water to provide raw water makeup to the Process Plant. Water balance modeling indicated that inflows to the CWP are typically 125 to 250 L/sec, depending on the Mine Year. This imbalance indicates that discharge of water from the contact water system is required in all but the very driest years, and geochemical analysis of the anticipated pit and WRD surface suggests that with the exception of the pre-production period and very early in the LoM, water that has come in contact with active mining facilities will not be suitable for release to the environment without some form of treatment. The water balance indicates that a water treatment with a capacity to discharge 180 L/sec of water is needed to maintain a stable volume in the CWP. The facility will be required during the first year of operations and will need to treat water throughout the LoM and into closure.

The CWP has been sized such that it can maintain sufficient surge capacity to contain the volume from an extreme wet season while still store sufficient water to supply the process with raw makeup during an extreme drought. This requires that the CWP be maintained within a fairly narrow operating range, which will require treatment of excess water on a regular basis.

The CWP is located in the Infirmes drainage below the CWRD and above the final footprint of the TSF. The CWP will be constructed with upstream and downstream dams to provide a storage capacity of approximately 1.28 Mm^3. The CWP will serve to store contact water and also isolate the footprint of the TSF from incidental surface water run-on during the frequent construction of the TSF raises. The area upstream of the CWP is within the ultimate footprint of the CWRD and would benefit from filling with waste rock to provide gravity flow from the CWRD ultimate footprint into the CWP. The maximum height of the CWP embankments was set at 13.25 m. This allows the CWP to fall within the Class C Dam Category of the French Guiana Commission des Grands Barrages (minister de l'écologie, 2015) namely;

Height >5 m and

Height^2 (in m) x $\text{Volume}^{0.5}$ (in Mm^3) < 200.

A culvert spillway from the CWP into the existing Topaze drainage will be provided under the haul road to the TSF. During the early LoM when contact water is expected to be relatively benign, the CWP will function as a sediment retention pond and allow discharge through the culvert spillway of water with low suspended sediments. Later in the mine life when contact water quality is expected to be degraded, the pond will be maintained at a low enough level that no water is discharged through the spillway culverts at events less than the 100-yr, 24-hr storm event.

After milling activities have ceased, the CWP will not be required to provide a surge volume and contact water inflows will be significantly reduced. During the closure activities, the culvert spillway under the haul road will be removed and the spillway lowered to allow the area of the former CWP to freely drain into the Topaze Creek drainage.

18.3 Tailings Storage Facility

18.3.1 Foundation Characterization

An SRK Geotechnical Engineer mobilized to the Project site from November 2015 through February, 2016 to oversee a geotechnical site characterization program for the TSF, CWRD, WWRD and plant site areas. The field program was primarily focused on the TSF, however, characterization of the foundation conditions for the CWRD, WWRD and plant site was also performed during this time.

The geotechnical field investigation was adjusted several times to account for changes in the TSF layout due to constraints identified after the start of the field program. Therefore, some drillholes and test pits originally targeted for the TSF area were ultimately located in the CWRD. The investigation of the TSF area was comprised of the following:

- Twenty locations were drilled with a Hanjin 6000 drill with a 1.5 m long HQ3 triple tube coring system. Six drillholes were advanced in the TSF footprint, six drillholes in the CWRD footprint, one drillhole in the WWRD, one drillhole in the plant site area and six drillholes were located outside of the currently defined TSF, WRD and plant site footprints;
- Four twin drillholes. In order to obtain additional geotechnical samples for laboratory testing, two drillholes were twinned in the TSF area and two drillholes in the CWRD;

- Thirty-four test pits. The test pits were completed to collect subsurface bulk soil data to complement the drilling data. Eight test pits were excavated in the TSF footprint, 15 test pits excavated in the CWRD footprint, five test pits excavated in the WWRD, two test pits excavated in the plant site area and four test pits were located outside of the currently defined TSF, WRD and plant site footprints; and
- Five shallow piezometers. Three shallow piezometers were installed in the TSF footprint, one shallow piezometer in the CWRD and one shallow piezometer installed outside of the currently defined TSF, WRD and plant site footprints.

In general, the materials identified as part of the field program within the TSF footprint were characterized in the following manner:

- Surficial Soils;
- Deposited Soils;
- Residual Soils;
- Weathered Bedrock (Saprock); and
- Bedrock.

18.3.2 Tailings Characterization

Three samples of ore were processed and provided to SRK as representative samples of the tailings to be produced at the Project. These samples were selected from three zones within the Pit, defined as Saprolite/Saprock, UFZ and LFZ. The UFZ and LFZ tailings behaved similarly in the column settling tests and required significantly less time to reach self-weight settlement equilibrium than the Saprolite. The saprolite tailings are estimated to constitute 11% of the total tailings mass, and the remaining 89% is expected to be an even blend of LFZ and UFZ tailings. Therefore, the properties of the UFZ and LFZ tailings were used to represent the tailings.

SRK performed a geochemical characterization to assess the Acid Rock Drainage and Metal Leaching potential of tailings for the Project (SRK, 2017a). The testing indicated that the tailings slurry will be alkaline when initially discharged into the TSF due to the alkalinity in the liquid fraction. However, due to the overwhelming net AGP, the tailings will likely generate acid over time if exposed to oxygenated conditions or metals could be leached to the environment.

18.3.3 Basis of Design

As part of the PEA, TSF-4 was selected as the preferred TSF location in the PEA because of its central location, existing disturbed ground and lower estimated capital costs. During the Project kick-off meeting in October 2015, it became apparent that the proximity of TSF-4 to the open pit could be problematic and SRK shifted the TSF approximately 500 m north of the original TSF-4 location. SRK was subsequently informed by SOTRAPMAG S.A.S. (SOTRAPMAG) that they did not have surface rights to Concession C01/32 at the northwest corner of the TSF embankment, and SRK developed a TSF layout that avoided Concession C01/32. After the capital costs were evaluated in more detail during the August 2016 Plenary Session, SRK developed a conventionally (slurry) deposited tailings in a TSF located approximately 500 m to the north that covered part of Concession C01/32. Because of the lower cost and changes in land ownership, Nordgold instructed SRK to proceed with the conventional TSF layout as the basis of the BFS.

SRK developed a TSF design that follows the French Guiana requirements for BAT, and was based on the following design and operating requirements:

- Production Rate: Mining and milling production at a rate of 4.6 Mt/y (12,330 t/d) (measured in dry tonnes) at steady state production, operating 12 months of the year;
- TSF Capacity: Tailings will be stored in a TSF sized to contain 56 Mt, constructed in four phases in approximate two to three year increments;
- Embankment. The tailings embankment will be constructed in a downstream manner using an initial starter embankment to minimize initial capital and subsequent downstream raises, and using material borrowed from within the impoundment and/or suitable excess overburden saprolite and waste rock borrow provided by the Mine;
- Stability: The tailings embankment must be stable and designed to the standards consistent with the hazard classification and modern embankment (dam) engineering practice;
- Density: To account for the rate of rise of the tailings in a valley configuration, SRK assumed an initially low average tailings dry density of 1,200 kg/m^3 for the first two phases and 1,250 kg/m^3 for the final two phases;
- Deposition: Conventional tailings slurry will be discharged from spigots located on the embankment to develop a tailings beach, and corresponding supernatant pool, away from the TSF embankment;
- Tailings beach slope: A beach slope of 0.5% away from the embankment was assumed for the design;
- ARD Potential. Based on SRK's understanding of the tailings ARDML issues, and in order to keep the tailings relatively saturated the TSF was designed without an overliner (secondary drainage system);
- Supernatant. Supernatant will be reclaimed and returned back to the mill using a barge system, which will provide a majority of the make-up water to the mill; and
- Spillway: A spillway will be constructed for Phase 4 and closure, sized to convey run-off from the PMP event.

18.3.4 TSF Design

The TSF design was based on the elements and design features discussed in the following sections.

Embankment Design and Embankment Type

SRK selected a downstream embankment construction technique for the Project. In downstream construction, the crest moves progressively downstream (or outwards) as the TSF is raised. While this configuration requires the largest embankment fill volume, it also typically provides the highest static and pseudo-static stability as the embankment fill is founded on competent foundation soils or bedrock (rather than tailings).

The tailings embankment will be constructed over four phases to minimize capital requirements.

Seepage Control

Seepage control for the tailings will be provided using a single 2.0 mm Linear Low Density Polyethylene (LLDPE) geomembrane over a prepare subgrade surface within the entire TSF impoundment. While High Density Polyethylene (HDPE) and LLDPE geomembrane are the most common lining materials, LLDPE geomembrane was selected due to its higher puncture resistance and greater elongation properties.

Leak Detection System

Given the capital costs and schedule impacts of installing a double liner system and a granular drainage layer, an "internal" Leak Detection System was not included in the TSF design. In the absence of a dedicated leak detection system, it is assumed that the proposed underdrain system will intercept the leaked supernatant and direct it to the underdrain sump. Water reporting to the underdrain sump will, depending on its quality, either be discharged to the environment or pumped back into the TSF.

Underdrain System

An underdrain system, comprised of a free draining granular material, will be installed within the TSF footprint in the area of any springs or seeps, to collect any groundwater. This underdrain will flow via gravity to a sump outside of the TSF footprint, to protect groundwater and minimize any uplift pressures on the geomembrane liner system.

Tailings Deposition and Supernatant Pool

Slurried tailings will be pumped from the mill to the impoundment via the Tailings Delivery Pipeline, and then to planned deposition locations (Deposition Points) via the tailings Deposition Pipeline using sub-aerial deposition. Deposition will be initially performed mainly from embankment deposition points to push tailings and entrained water away from the embankment and simultaneously establish deposition cycles that optimize the creation and maintenance of a well-drained beach with a positive gradient to the southwest (i.e., away from the embankments).

The design of the Tailings Delivery Pipeline, Deposition Points, Reclaim Water Pipeline has been done by others.

Tailings Consolidation Settlement

SRK estimated the tailings density and time to reach 90% consolidation post-deposition for the three ore types (LFZ, UFZ and Saprolite) using FSConsol parameters (FSConsol, v2007). For the purpose of volume calculations used in the design and layout, SRK used an average tailings dry density of 1.2 tonnes per cubic metre (t/m^3) for the first two phases (upper third) and 1.25 t/m^3 for the final two phases (lower two thirds) to represent the three tailings materials.

Overdrain System

Based on the consolidation modelling results and time to reach 90% consolidation, and potential ARDML issues, an Overdrain System was not considered necessary. By maintaining the tailings in a saturated condition and removing the oxidation potential, the onset of acid generation is essentially prevented. By contrast, maintaining the tailings in a saturated condition leads to higher liner leakage rates due to imperfections or defects in the geomembrane.

Stability Analysis Results

SRK identified critical stability sections for the North and South embankments, typically located where the embankments are at the maximum starter and ultimate height, and where the soft foundation soils (Deposited and Residual) have the greatest thickness. An additional critical stability section was identified through the South Embankment abutment, with a sliver fill placed to reduce the existing slope to 2.5H:1V to allow for the installation of the geomembrane liner.

The material properties used in the stability analysis are discussed below:

- Tailings: An effective shear strength of 30 degrees and 0 kPa was estimated from published values in the literature and SRK's past experience;
- Engineered Fill: This may consist of either local Saprolite cut, excess Saprolite overburden or excess Coarse Waste Rock (CWR) overburden. An effective shear strength of 35 degrees and 0 kPa was estimated from published values in the literature and SRK past experience;
- Deposited soils: An effective shear strength of 25 degrees and 0 kPa with a constant undrained shear strength to the effective vertical stress ratio of 0.3 was obtained from five triaxial tests completed with samples of Deposited soil from different locations within the TSF and WWRD areas.
- Residual Soils: An effective shear strength of 29 degrees and 0 kPa with an S ratio of 0.5 was obtained from triaxial tests completed with one sample from the TSF area, two samples from the WWRD area and one sample from the plant site; and
- Bedrock: An effective shear strength of 45 degrees and 100 kPa was estimated from published values in the literature and SRK's past experience.

In order to achieve acceptable static and pseudo-static FOS values, the top 5 m of the Deposited and low shear strength Residual soils needed to be removed under the North and South embankments, and the South Embankment abutment. Similarly, for the purpose of volume calculations used in the design and layout, SRK assumed that 5 m of existing foundation soils (Deposited and low shear strength Saprolite) would be removed from within the TSF embankment limits.

Water Balance

SRK developed a site-wide water balance model that tracked water and solids flows during the life of the Project. The site water balance model considered the mine pit, Process Plant and Mill, TSF, Ore stockpiles, CWRD, WWRD and Contact (Raw) Water Pond. As part of the site water balance GoldSim model, SRK included a water balance for the TSF that incorporated the four TSF phases. Water inflows in the model included precipitation captured on the exposed lined area, exposed tailings, or open water areas; run-on from un-diverted or partially diverted watersheds; water in the slurry released at placement; and releases from consolidation of the tailings. Outflows used in the TSF water balance included evaporation from the exposed dry tailings beach, wet tailings beach and open supernatant pool; water temporary and permanently entrained in the tailings; decant from the supernatant pool for reclaim to mill; and treated discharge from the pool.

Slurried tailings from the Process Plant and Mill were deposited into the TSF, and excess solution was reclaimed from the supernatant pool and returned to the Process Plant and Mill as makeup. The model assumed non-contact water generated by run-off from the surrounding hillsides was diverted around the TSF and discharged to the west and north of the TSF.

The water balance estimates that approximately 150 L/sec of water is required to produce the tailings slurry. After water gained through ore moisture is accounted for, a makeup demand of 140 to 145 L/sec is required. The majority of this demand is satisfied by the TSF, i.e. typically 180 to 120 L/sec. The remainder of the demand is provided by water stored in the CWP or precipitation captured in the Plant and Mill bund area and made available for raw water makeup.

During the first phase of the TSF development, the precipitation captured within the lined TSF limits appears to roughly balance the losses in the system, and water treatment from the TSF is not required during the first phase of the TSF. However, once the second phase of the TSF expands the footprint of the TSF to 86 Ha, the model simulations indicated that excess water accumulates in the system and treatment and discharge of water from the TSF supernatant pool is required at a rate of 140 L/sec to maintain the supernatant pool level. It should be noted that the amount of treatment required on a monthly average basis varies significantly in response to high rainfall events.

Diversion Channels, Downchute and Spillway

To support the surface water management design, SRK developed a hydrology model to estimate the peak flows from the PMF from the PMP, the 1:100, and the 1:10 year storm events for the following surface water diversion structures:

- TSF diversion channels on the south side of the TSF impoundment to intercept and convey surface water from the hillslope above the TSF around the TSF. To accommodate the design flow within the limited channel width available, the TSF Diversion Channels were designed using a two-step approach. The TSF Perimeter Diversion Channel was sized to convey the 24-hour, 1:100-year return period event, with four Secondary Discharge Channels spaced evenly along the perimeter, constructed perpendicular to the TSF Perimeter Diversion Channel and sized to convey the 1:100-year storm event from the main diversion channel directly into the TSF;
- The TSF was designed with a Downchute Channel to discharge surface water from the TSF Perimeter Diversion Channel and discharge the surface water into the North Sedimentation Pond. The Downchute Channel is intended to be constructed and operated between Phases 1 through Phase 3 for flows up to the 1:100-year peak event that report to the Phase 1 through Phase 3 TSF Diversion Channels. The Downchute Channel will be abandoned once the TSF Phase 4 construction is complete; and
- The Closure Spillway was designed to be constructed in Phase 4, once the TSF Embankment has been constructed to its ultimate elevation. During the Phase 4 Operational period, the Closure Spillway will convey flows from the TSF Perimeter Diversion Channels. The spillway also functions as an emergency spillway for PMP level flows on the active TSF. Post-operations, the Closure Spillway will convey a combination of surface water flows from the reclaimed surface of the TSF and run-on reporting to the TSF Perimeter Diversion Channels. During closure activities, once the TSF surface has been reclaimed, the invert of the Closure Spillway will be lowered to the level of the TSF closure cover to allow free draining discharge from the TSF.

Construction

Construction for the TSF will be performed in four phases to reduce the initial construction costs. In general, the construction will include foundation preparation, placement of compacted fill and installation of a geomembrane liner, as follows:

- Foundation Preparation: The TSF Embankment foundation will be prepared prior to construction. Any vegetation and approximately 5 m of unsuitable foundation material (Deposited soils and low shear strength Residual soils) will be removed, and the surface scarified, moisture conditioned and recompacted prior to placement of Engineered Fill;
- Underdrain System: Springs and seeps identified previously and during construction activities will be collected in an underdrain system that will gravity flow to a point north of the TSF footprint;

- Engineered Fill: The TSF footprint will be re-graded and embankment constructed to achieve the required grades. Engineered Fill generated from cuts within the TSF and waste material imported from the Mine will be used to achieve the final fill grades shown on the drawings. Depending on the gradation of the material, it will be placed to a method or performance based specification;
- Impoundment Fill. Within the TSF Impoundment limits, 2 m of Impoundment Fill will be placed in one lift over Deposited soils to provide a suitable working surface for geomembrane liner deployment and installation;
- Liner System: A 2.0 mm LLDPE geomembrane liner will be placed over a prepared surface within the entire TSF Impoundment limits.

CWR (from mine overburden) will be used for underdrains, riprap and Engineered Fill, and Saprolite (from either local cut or mine overburden) will be used for Engineered Fill and Impoundment Fill. As the majority of pre-production excavation will be Saprolite during Phase 1 construction, SRK has estimated that there is only a sufficient quantity of CWR available for underdrain construction and that the Engineered Fill for the Phase 1 embankment will be comprised of Saprolite. The Saprolite used in the Phase 1 construction will be sourced from cut areas within the Phase 1 TSF, to increase the TSF storage capacity and reduce the risk that the mine cannot schedule sufficient Saprolite material during the pre-production mining period. During production, the CWR produced by the mine will increase, resulting in approximately 50% of the embankment being constructed of CWR for Phase 2, and 90% for Phases 3 and 4.

19 Market Studies and Contracts

19.1 Contracts and Status

Markets for doré are readily available. Gold markets are mature, global markets with reputable smelters and refiners located throughout the world. Demand is presently high with consistent prices for gold during recent times.

The 36-month average London Bullion Market (LBM) gold price through February 2017 is US$1,215/oz (SNL Metals & Mining website, February, 2017) and the February 2017 monthly average spot close price is US$1,257. For the purposes of this report US$1,200/Au oz has been used for reserves and US$1,250/Au oz has been used for economic modeling.

Nordgold does not have any material contracts that may pose a financial liability at this time (2017). It is assumed that gold production will be sold to a generic European precious metal refiner.

20 Environmental Studies, Permitting and Social or Community Impact

20.1 Site Visit

SRK's environmental specialist and QP, Mr. Mark Willow, conducted an initial reconnaissance of the Project site from April 1 to 3, 2014. The visit included an overview tour of the proposed mine area, including, but not limited to the location of the open pit (open cut), potential processing areas, and potential TSF areas. Given the early exploration and development phase of the Project at that time, no physical infrastructure of the proposed Project was available for inspection of this greenfield Project. There was, however, considerable evidence of non-regulated artisanal mining occurring within the proposed Project footprint. This issue is discussed further below. No subsequent site visits have been conducted by SRK personnel for the purpose of environmental inspection or review.

20.2 Known Environmental Liabilities

The Project area is an intermittently active exploration and alluvial mining property centred in dense tropical rain forest of French Guiana. Exploration activities have required access road and drill pad construction, trenching, water management features, as well as erection of worker encampments. Environmental liabilities resulting from previous and ongoing exploration and mining activities are fairly limited due to the high precipitation and rapid natural revegetation that occurs in the rainforest. Holders of exploration permits are required by law to reclaim worked areas, control stormwater, and potential sedimentation of downstream surface water resources, and since January 2006 are strictly prohibited from using mercury for mineral beneficiation. These conditions are monitored closely by the regulatory authorities.

Illegal artisanal placer mining that occurs over much of the Project area has disturbed considerable land area, and continues to impact local surface water resources through sediments release and water contamination.

20.3 Environmental Study Summary

In addition to information gathered during the site visit, SRK was provided a copy of the *Montagne d'Or Gold Project Environmental Scoping Study* prepared by WSP Canada Inc. (WSP), dated September 2015. WSP (2015) provides an overview of the environmental and socio-economic issues for the Project, as well as a preliminary indication of the positive and potentially negative impacts associated with the planned operation, which is intended to provide direction for the continuing environmental assessment process, and guide the environmental authorities with the information required to determine the range of information and degree of detail needed in the formal impact assessment. The study area, as defined for the environmental studies purpose, includes concession n° 215 – C02/46, the access route from Saint-Laurent-du-Maroni, and a 500 m wide buffer zone on each side of the access corridor.

The following are brief summaries of some of the key environmental and social issues presented by WSP (2015) as well as other relevant technical studies prepared for the BFS. It is not the intent of this section to reiterate the results of the environmental scoping study in their entirety. The reader is, therefore, directed to WSP (2015) for more detailed information and analysis.

20.3.1 Biological Reserve and Resources

Baseline data on the fauna, flora and habitats covering the Project area and presented by WSP (2015) were obtained from a review of the published literature, existing databases, and preliminary field inventories conducted in 2014. Many other field campaigns were conducted by WSP and its consultants in 2015 and 2016. These inventories concentrated on floral taxa and terrestrial and aquatic faunal taxa, mostly in the vicinity of the mining site and Camp Citron as well as along the access road, and were timed to coincide with wet season and dry season conditions at the site.

The Project is located between the two sections of the RBI of Lucifer Dékou-Dékou, in a space designated as RBD. The Lucifer and Dékou-Dékou massifs are home to two floral assemblages rare in French Guiana: the sub-montaneous forest on lateritic bauxite hardpan, and the forest on 400 to 500 m slopes. They shelter some fifty floral heritage species and three nationally-protected species. This heritage value led to the creation in 2012 of the Lucifer and Dékou-Dékou RBI, the first such reserve in French Guiana and the largest in French jurisdiction. Within the RBI, any direct human intervention that could modify the functioning of the ecosystem is prohibited. The only authorized sylvicultural measures are those eliminating exotic or invasive species and the securing of trails and roads bordering or crossing the reserve.

While the Project itself is located in portions of a RBD, mining activity is permitted under certain conditions. This exception was established to take into account historic exploration and exploitation of gold resources in the area, as well as the presence of potentially significant mineral deposits at the foot of the Dékou-Dékou massif.

The field inventories conducted from 2014 to 2016 in the study area found 1558 species of plants and 505 species of terrestrial vertebrates (299 birds, 35 mammals, 63 bats, 58 amphibians, 50 reptiles). The aquatic fauna surveyed comprised 52 families of macro invertebrates and 41 species of fish.

The highest diversities of plants and birds were found in the steep slope old-growth and dense evergreen forests. The richest community of amphibians, almost half the number of species observed, was found in the alluvial mining areas and their interfaces with the second-growth forest.

On the basis of the principle "avoid-reduce-offset", optimization measures of the Project have been developed in order to avoid impacts on biodiversity, including the elimination of the WRD to the northeast of the pit in order to preserve the wildlife migration corridor. Measures to reduce the impact will be also prescribed in the impact assessment study. In addition, a compensation program tailored to the scale of the Project and the challenges of biodiversity is underway with the local partners in order to compensate for residual impacts on biodiversity. For example, the restoration of the illegal gold mining sites close to the future mine, and support for the management and protection of the RBI Lucifer – Dékou-Dékou are important objectives for the Project.

As part of the biological baseline data collection program, fish tissue samples were analyzed for mercury content using an AMA 254 spectrophotometric absorption mercury analyzer. A total of 245 specimens were sampled across several locations. Stations located upstream and within the deposit area did not produce any specimens with concentrations greater than the World Health Organization (WHO) limit of 0.5 μg Hg/g. In contrast, the downstream stations in the flats yielded 39 samples above the WHO limit, as would be expected in locations where illegal artisanal mining which uses mercury was occurring.

20.3.2 Threatened and Special Status Species

Article L. 411-1 of the French Environmental Code (FR197 – consolidated version of 2010) strictly protects wild species of plants and animals listed by Ministerial Order. These species cannot be captured, transported, intentionally disturbed or commercially exploited. These prohibitions can extend to the destruction, degradation and alteration of the habitats of these protected species. A total of 110 nationally protected species were recorded on the Project site. Of these protected species, 100 were bird species, including three species with protected habitat, seven were mammalian species, and three plant species.

The site also hosts five plant species new to French Guiana and seven other plants of interest (rare or endemic), as well as two fish species rare and endemic to French Guiana, present on the mountain creeks.

20.3.3 Air Quality

The field program to establish baseline air quality at the Project site took place during the dry season, from 9 to 15 October 2014. WSP (2015) concluded that the overall air quality was good, given the lack of human activity in the area and the dense forest cover. As a result, the sensitivity regarding air quality will likely be high, especially since the RBI LDD including the Dékou-Dékou massif, to the south of the Project, and the Lucifer massif, to the north of the Project, must be preserved.

The Project will be subject to the guidance and recommendations on the use of BAT entitled *Non-ferrous Metals Industry* (European Commission, 2009). This document lists the best practices regarding data collection and pollution minimization of precious metal treatment processes.

20.3.4 Cultural and Archeological Resources

Following the issuance of the prospecting authorizations by the Ministry of Culture (Order DAC-SA n^{o}. 17, of August 7, 2014 and Order DAC-SA n^{0} 2016-62, of November 10, 2016), two archeological prospecting campaigns were conducted on the Project site. Institut National de Recherches Archéologiques Préventives (INRAP) instructed specifically to explore Native Amerindian archaeological sites while the consultants, Mine & Avenir, investigated contemporary archaeological sites (panning and mining activities).

Pedestrian survey campaign has ended with the discovery of 47 proven sites attributed to the pre-Columbian period, which demonstrates a strong archaeological potential of this region. The Lidar revealed the presence of fifteen "crowned mountains" including 10 sites that are spread over an area of about 40 km² around the future Project. This number reflect social behaviors that led to such developments and reinforces the unique heritage character of the region. These features generally involve the "construction" of deep ditches encircling some of the summits of hills in the forest, within which collections of relics (pottery and stone tools), along with radiocarbon dating, suggest that Amerindian groups had occupied these locations. To the extent practicable, these locations are avoided by the mine plan.

The investigations carried out by Mine & Avenir have shown the importance of human occupation related to gold mining in the Paul Isnard sector which has developed almost continuously since 1870, with a rich mining history and a very strong human footprint. This footprint translates the presence of many mining villages sometimes with their cemetery, at the bottom of valleys and mountains-foot of the Dékou-Dékou. In general, investigations showed the highly degraded nature of different remains by the successive phases of mining activity (destruction of the remains of old works, but also of habitats associated with mining, etc.). Among these remains, the most important heritage sites exist along the route between the former mining village of Paul Isnard and its cemetery, both areas almost entirely destroyed by various earthworks, and the cemetery of the Placer Enfin along the Infirmes creek. This cemetery (approximately 20 graves) remains relatively intact. Under the current mine plan design, however, intact preservation of this feature is probably not possible given its location within the footprint of the planned TSF. This issue will be evaluated as part of the environmental impact assessment of the Project.

20.3.5 Land Use

In the Project area, most land (including the access road between SLM and Citron Camp) consists of wet lowlands forest. Near Saint-Laurent-du-Maroni, slash-and-burn farming is performed on small plots along the road toward the Croisée d'Apatou. This road crosses the State's private forest. This forest, and the road up to Croisée d'Apatou, is managed by the National Forestry Office. A few forest exploitation sites, a sawmill, and at least three laterite quarries are located near the access road corridor.

20.3.6 Hydrogeology (Groundwater)

From 2014 to 2016, a baseline groundwater program was conducted at the site as part of the ongoing *Environmental Impact Assessment Study* made by WSP. Additional site investigations and hydrogeological modeling was conducted by SRK (2017) as part of the technical studies supporting the BFS. Data were collected through a network of 15 piezometers installed around the Project area.

The results of these programs suggest that the hydrogeology of Montagne d'Or is largely controlled by site topography and three primary geologic units: a low-permeability saprolite cover; a higher-permeability saprock layer; and a variably-permeable fractured bedrock. Recharge to the groundwater system appears to occur over a large area as infiltration through the saprolite, except along drainages where the saprock and bedrock are exposed. Surface run-off and shallow groundwater from the drainages where saprock is exposed will be captured and diverted around the proposed open pit. However, groundwater in bedrock, and in faults and joints within the bedrock, will report to the open pit and require in-pit sumping during operations.

At the end of mining, the open pit will fill with a combination of groundwater, surface run-off, and direct precipitation. Once the pit lake reaches the rim, it will discharge into existing drainages, since the pit lake water quality is currently anticipated to be suitable for discharge.

The hydrogeological model developed by SRK (2017) predicts that drawdown within bedrock below the saprolite may propagate south past the boundary of RBI (RBI). As a result, a potential minor reduction in groundwater discharge to Apollon Creek (less than one half of 1 percent) at the end of mining is possible., This reduction in groundwater discharge would be undetectable in field measurements.

Groundwater quality monitoring shows that the waters are slightly acidic with low major element concentrations, a bicarbonate calcic profile, and no contamination (i.e., no elevated concentrations of regulated constituents).

20.3.7 Hydrology (Surface Water) and Water Quality

(2015) focused their baseline characterization efforts on the creeks and drainages near the proposed Project site that had the potential of being impacted by the proposed operations. Infirme Creek (in concession C02/46) drains the majority of the surface waters flowing from the Dékou-Dékou Massif's plateau towards the north. This watercourse flows into the Reine Creek. The Project primary access road also crosses numerous drainages from Croisée d'Apatou to Citron, which will require detailed evaluation and engineering of crossing structures (bridges, culverts, hollow logs, etc.).

Based on the extrapolation from regionally available climatological data, SRK (2016b) was able interpolate rainfall depths and intensities at various frequencies and durations for use in the design of structures for the Project. 24-hr duration frequency storms predicted by this analysis range from 117.2 mm (2-year storm) up to 628 mm (500-year storm), indicating an area of substantive annual rainfall requiring proper stormwater controls. The municipality of Saint-Laurent-du-Maroni, in which the Project is located, does not currently have a Flood Risk Management Plan, nor any maps of flood zones for the creeks in the WSP (2015) study. During heavy rainfall, the valley bottoms flood quickly given the small catchment areas and high rainfall in the region.

During initial site reconnaissance, numerous mudslides were observed to the east and west of the Project, indicating land movement which could affect surface water flows and water quality in Roche and Violette creeks to the west and Beaurivage Creek to the east. In addition, active artisanal mining operations were observed discharging heavily sediment-laden water and processing waters to the adjacent drainages. As part of the baseline monitoring program, a number of in situ surface water sampling locations were established by GéoPlusEnvironnement from which regular water quality monitoring is being conducted. Initial monitoring results find the following general characteristics for site surface waters:

- Low hardness;
- High TSS concentrations;
- Basic pH indicating mineralization of the water; and
- Presence of mercury and cyanide (from illegal artisanal mining).

The site-wide water balance conducted by SRK in 2016 indicates that the operation will need to collect, treat, and discharge excess water. These discharged waters must achieve "good ecological and chemical status" as defined in the Law on Water and Aquatic Environments (LEMA) N°2006-1772 of 30 December 2006 governed by the DCE.

20.3.8 Geochemistry

Following indications in the PEA of the potential for substantial volumes of sulphidic waste rock, SRK recommended construction of a NAG-PAG block model that would support operational material handling and segregation of mine waste blocks predicted to have a high potential for ARDML. To provide a dataset that would allow valid statistical analyses to support the model and provide comprehensive geochemical characterization of the primary waste rock types, SRK geochemists assisted by Columbus geologists collected 451 samples of drill core at site. All core samples were submitted to BV for static geochemical tests that included acid-base accounting (with sulphur and carbon forms) and multi-element analyses. SRK also conducted comprehensive geochemical testing on the detoxified master composite from the metallurgical program. Details of the geochemical testing program on waste rock and tailings along with results and interpretations are described in SRK (2017a).

Based on results of the static program, subsets and composites were generated for additional testing that included column leach tests, single addition NAG tests, KNAG tests, and mineralogical analyses. Data generated from the testing program were used to construct geochemical models for the purpose of providing predictions of long-term water quality of drainage that will be produced from tailings, waste rock, and the pit lake. Descriptions and results of the predictive geochemical models are presented in SRK (2017b).

The following is a high-level summary of the geochemical characterization test work. The waste rock, low grade ore, and tailings management will be subject to the guidance of the BAT- Management of Tailings and Waste-rock in Mining Activities (MTWR, 2009) and will likely follow recommendations from the draft Management of Waste from Extractive Industries (draft MWEI, 2016) and will comply to PAG waste storage French regulations.

Waste Rock

ABA was carried out on the waste rock samples to assess the balance of acid generating sulphide minerals and acid neutralizing carbonate minerals. The results of the static testing program suggest that bulk of the waste rock generated during operations (±55%) could be classified as PAG. Approximately ±30% of the material is likely non-PAG (non-acid generating, NAG), with the remaining fraction (±15%) classified as uncertain in nature.

In addition to the ABA testing, a series of kinetic tests were completed on waste rock which provided some contrary predictions to those observed in the static test work. The KNAG data indicate that only the Felsic Tuff and the Lapilli Tuff are PAG due to the apparent encapsulation of sulphides in quartz and other silicate phases which renders the sulphide minerals unreactive. This is a significant finding that indicates that the mass of acid generating waste rock is considerably less than indicated by the ABA results, which has important implications for waste rock management plans.

Leaching tests were also conducted on the waste rock samples per CEN/TS 14405 methodology. The results from the 43 column tests indicate that the waste rock materials have the potential to leach barium (Ba), chromium (Cr), copper (Cu), nickel (Ni), lead (Pb), antimony (Sb), and zinc (Zn) at elevated concentrations. No mercury (Hg) was detected in the samples. In light of these results, the potential for leaching metals remains a concern at this stage, and will need to be considered during detailed design and construction of the mine.

At closure, the recommended reclamation option is to cover and revegetate the WRDs to stabilize the dumps, minimize infiltration of water and oxygen, and construct drainage pathways that direct run-off away from the dumps. Isolation of the identified PAG rock from water and oxygen early on during operations is highly recommended, with the objective of precluding the accumulation of acidic pore water in the dump that could persist into closure and create a scenario in which draindown of acidic, metal-laden water is released to the environment slowly over time.

Low-Grade Mineralized Material

For the most part, the low-grade mineral samples tested tend to be classified as potentially acid generating and will need to be managed as such during operations. It is currently anticipated that this material will be processed toward the end of mine life, and therefore not expected to create post-closure ARDML.

Tailings

Several composite tailings samples, following cyanidation and detoxification, were characterized geochemically in parallel with the liquid fraction supernatant. The detoxified tailings were found to have 1.2% residual sulphide with a total carbon level below detection at <0.01%, indicating that the tailings solids are likely to be net acid generating. The supernatant will initially be alkaline when first discharged to the TSF, and should aid in buffering system. Metallurgical testing indicates that detoxification using the INCO SO_2/air procedure is successful in reducing cyanide species to below effluent limits for eventual discharge to surface waters post closure.

Pit Lake

Hydrogeological modeling in the area of the open pit predicts that, at closure, the pit will fill with water and start overtopping in approximately 6 years. This rapid influx of water will have the effect of introducing a large volume of relatively clean water in a short time period, which results in reasonably good pit lake water chemistry sustained into the future. The risks of creating a low-quality pit lake post closure, due to the exposure of this water to the pit wall lithologies, are minimal.

20.4 Socio-Economics

Covering an area of 83,846 km², French Guiana has 22 towns in 4 communities of communes and 19 cantons. The closest community to the Project site, the town of SLM, has a Local Urbanism Plan (PLU), approved on 8 October 2013, which contains provisions affecting the development of mining in the region, including:

- Develop a higher education sector with the creation of a training centre;
- Enable the development of mining in a manner that is consistent with the protection of the natural environment requirements, the preservation of agricultural and forest areas and remarkable ecological environments;
- Reconcile the imperatives of urban and economic development and the preservation of the natural environment, heritage and identity, protecting specific areas harboring endemic species such as the ZNIEFF Dékou-Dékou (high points) and Lucifer massifs (high points) and special and remarkable landscapes participating in the area's identity, including Chutes et Crique Voltaire; and
- Qualify the economic river entrance to the city, for the development of river frontage at ports.

Moreover, the Project sector is classified as an Nf zone (natural protected area due to the presence of wooded areas where the forest's vocation is recognized) under the PLU; the regulation authorizes the extraction of materials subject to compatibility with SDOM. The latter classifies Montagne d'Or as a zone where mining activity is permitted, but under constraints, given the environmental sensitivity. This should not, however, materially impact mine permitting and development.

20.4.1 Gold Mining

Gold mining is fairly well developed in French Guiana, with 42 mining claims and 52 operating licenses on record in 2013. For 2013, French Guiana reported the production of 1.3 t of gold, generating a regional mining tax of €550,000, a departmental and communal fee of €200,000, and a fee to ONF of approximately €200,000. However, the industry continues to be plagued by illegal artisanal mining, which, according to the ONF, consisted of 774 sites in 2013, for an estimated annual production of 10 to 12 t of gold.

20.4.2 Native American and Bushinengues Communities

It is important to note that there is, in French Guiana, an ethnic identity that transcends nationalities and administrative boundaries. The Maroni River is also the backbone of a territory stretching along its Guyanese and Surinamese banks, where from the middle to the bottom Maroni, home to Native American ethnic groups (Arawak and Kali'na), the Bushinengué ethnicities (Aluku, Saramaca, Paramaca, and Djuka) and Creole ethnicities issued from slavery.

In French Guiana and neighboring Suriname, Bushinengues (meaning 'people of the forest') are identified, in part, as descendants of former African slaves who escaped in the late seventeenth and early eighteenth centuries from former Dutch plantations. Their communities were the result of a growing movement amongst slaves to escape enslavement and establish independent communities in the forest. While a significant portion remained in Suriname, the majority fled to French Guiana.

The Aluku is one of the Bushinengues ethnic groups in French Guiana, who, toward the end of the eighteenth century, settled alongside the riverbanks of Lawa Maroni, which now forms the border between French Guiana and Suriname. There were at least two other groups of escaped Africans in the area, Saramaka people and the Ndyuka people, who eventually assimilated with the Aluku to form a new ethnic group.

In the late eighteenth century, the Aluku occupied the region of Saint-Laurent-du-Maroni, Apatou, Grand-Santi; the largest piece of the territory still occupied is called Fochi-ké (First Cry), better known as Aluku, located in the region of Maripasoula, consisting of the municipalities and city of Maripasoula and the capital city of Papaïchton, and the traditional villages of Kormontibo, Assissi, Loca, Tabiki, and Agoodé, in French Guiana, as well as the Cottica, in Suriname. There is also a very large Aluku population in Saint-Laurent-du-Maroni, Cayenne, Matoury, and Kourou.

According to IFC performance standards 7 – Indigenous Peoples, a thorough evaluation of the natural resources usage by indigenous peoples will be launched in order to assess impacts of the Project on natural resource-based livelihoods and compensate for such impacts.

20.5 Environmental/Social Issues and Impacts

20.5.1 Stakeholder Engagement

As part of the ESIA of Montagne d'Or, an initial stakeholder consultation mission was performed by WSP between September 15 to 19, 2014, in Cayenne and Saint-Laurent-du-Maroni. The initial consultations consisted, for the most part, of individual interviews with departmental authorities and technical services, in addition to environmental NGOs and economic organizations. These meetings highlighted potential environmental and social issues associated with the Project and for which a public, professional or legal concern may arise.

Results from these initial consultations are presented in the document, *Stakeholder Consultation* (WSP, 2014), which includes the general consultation approach, the organizations and individuals that were engaged, and the key concerns and expectations that were identified. Those findings are summarized in the following sections.

20.5.2 Principal Stakeholder Issues

WSP (2014) summarizes the main issues and concerns expressed by stakeholders during a first series of consultations which took place in September 2014. The purpose of identifying the issues and concerns is:

- To guide the Project's continuing environmental assessment process, specifying the factors which will require specific attention; and
- To focus the Project early on in the design process, specifying the environmental and social considerations needing to be taken into consideration to ensure compliance with the regulatory framework and to avoid significant effects on the natural or human environments.

While WSP (2014) classifies these issues and concerns according to three levels of significance, only those identified as Level 1 are presented herein. Level 1 is given to issues which are subject to standards or regulatory aspects, which are deemed important for the Project's acceptance by stakeholders, and which could generate significant environmental or social impacts. The issues noted as Level 1 are summarized below as follows:

- Biodiversity and natural areas:
  - Integrity of the Lucifer Dékou-Dékou/RBI;
  - Protection of flora and fauna and quality of biological inventories;
- Stakeholder consultations:
  - Proactive and transparent communication;
- Economic development:
  - Local and regional jobs and economic spinoffs;
  - Supply of energy;
  - Training of qualified local workforce;
- Fight against illegal gold mining:
  - Contribution to the fight against illegal gold mining;
- Pollution prevention:
  - Sound environmental management;
  - Prevention of pollution and industrial risks, including those related to the eventual use of cyanide;
- Protection of watercourses and catchments:
  - Protection of catchments (Mana, Sparouine);
- Safety and crime:
  - Controlling traffic on the Project access road;
  - Securing the mine site; and
  - Workplace health and safety.

Level 2 issues and concerns have no applicable regulations, but are deemed important for certain stakeholders or could produce environmental or social impacts, while Level 3 is used to classify issues for which it is generally desirable to avoid negative environmental or social impacts.

20.5.3 Project Advantages

The Project is likely to generate positive effects which must also be taken into account during the environmental assessment and design processes. In general, the main advantages of natural resource projects are the creation of direct and indirect jobs, the stimulation of companies who supply products and services, and increased fiscal benefits related to economic stimulus and royalties. The stakeholders consulted in September 2014 identified additional opportunities related to the Project's specific situation:

- Fight against illegal gold mining in the region;
- Reduce gold pillaging from French Guiana;
- Stop discharge of mercury into the environment (its use has been banned since 2006);
- Stop damage to the RBI LDD, to the bottom of valleys and to creeks, as well as wildlife poaching by illegal miners;
- Improve development of mining industry in French Guiana;
- Increase revenue to the region;
- Train qualified workers in various technical and professional sectors; and
- Restoring degraded sites in the RBI LDD.

20.6 Environmental and Social Management Planning

WSP (2015) includes recommendations on avoidance strategies, mitigation alternatives, compensation, and monitoring measured in order to ensure compliance with the respective regulatory frameworks for the Project and the environmental resources. These actions and activities are typically detailed in Environmental Management Plans (EMPs) and Social Management Plans (SMPs) developed as a direct result of the ESIA process. Unfortunately, the ESIA process has not yet been initiated, and is awaiting the completion of the BFS design basis for the Project (presented herein).

While the development of the site-specific EMPs and SMPs has not yet been initiated, the principal areas of potential impact, and thus the focus and framework for Project environmental and social management planning will include:

- Environment, Social, Health and Safety (ESHS) Training Plan;
- Pollution Prevention Management Plan;
- Water Management Plan;
- Waste Management Plan;
- Hazardous Material Management Plan;
- Biodiversity Management Plan;
- Erosion Control and Re-vegetation Management Plan;
- Labour Management Plan;
- Community Health, Safety and Security Management Plan;
- Community Development Plan; and
- Stakeholder Engagement Plan.

These plans will be developed during the ESIA development and review process.

20.7 Project Permitting Requirements

WSP (2015) provides a preliminary identification of the regulatory elements to which the Project is subject, based on information currently available on the Project. The intent of this section is not to reiterate that information herein, but rather provide a high-level summary.

In 2012, the National Government of France approved new legislation promoting the development of the mining industry French Guiana. The legislation, known as the *Schéma Départemental d'Orientation Minière* (SDOM) was created with the objectives of encouraging economic development of the mining industry in French Guiana while protecting its environment. To accomplish these objectives, the SDOM provides increased security of land tenure, clarifies mineral development guidelines and environmental conditions and restrictions, and assigns lands in French Guiana zones that define limitations on mining activity:

- Zone 0: Banned for exploration and mining.
- Zone 1: Open to aerial surveys, underground mining authorized subject to conditions.
- Zone 2: Open to exploration, underground and open pit mining authorized subject to conditions.
- Zone 3: Open to exploration and underground and open pit mining.

Most of the Project concession areas, including the Montagne d'Or gold deposit, lie within Zone 2. Some of the conditions for mining in Zone 2 include:

- Demonstration of a viable mineral deposit;
- Completion of an Environmental Impact Study and Reclamation Plan; and
- Possible additional reclamation or environmental investigations, as may be required for the public interest, on or off site.

In addition to the land restrictions presented by the SDOM, the Project is located adjacent to a nature reserve, the RBI Lucifer Dékou-Dékou, managed by the ONF [French National Forestry Board]. Its Management Plan from the ONF is yet to be developed so there is little guidance or decisions regarding the use of land and allowable activities within the reserve. The boundaries of this reserve overlap 4 of the 8 Project mineral concessions; however, only one of these concessions is important to the Project. Since these concessions already exist, and there has been continued exploration and mining activity in the area for over 100 years, the ONF has agreed to create several zones within the reserve boundaries where mining is permitted. The Montagne d'Or deposit itself is within a zone where open pit mining is permitted and the outer limit of the open pit is located approximately 440 m from the reserve boundary.

It should be noted that a separate mineral concession, Concession 102 ("01/32"), on which the proposed TSF is partially located, is not currently owned by Nordgold. However, according to correspondence from the Ministry of Economy and Finances – General Secretariat of Urban Planning and Nature Development, dated February 2, 2017, the previous owner, Tanon and Co. has not requested an extension of the gold mine license as of December 31, 2016. As such, the area of Concession 102 will technically be open as of January 1, 2019, and available to Nordgold to acquire. A mining title is generally not required for infrastructures like the TSF, and Nordgold is currently negotiating with the ONF land use conventions based on the new layout presented herein.

A Mitigation and Rehabilitation Plan (MRP) is required for the operation to describe the mitigation measures for impacts on the environment, as well as final rehabilitation measures to be employed at the end of mine life. The MRP must include the calculation of, and mechanism for the operator to provide a financial guarantee to cover the mitigation and rehabilitation costs. The operator is authorized to set up a provision for the site rehabilitation in accordance with the provisions of article 258 of the Mining Code.

If, on completion of the exploration and/or exploitation works, the operator does not voluntarily execute the obligations agreed to in the MRP, the Mines Authority can seek court-ordered confiscation of the corresponding provision for rehabilitation set up by the holder. If the value of the guarantee or provision thus confiscated is not sufficient to cover the costs necessary to return the site to its original state, the Mines Authority may entrust execution of the pending work to a third party. The costs incurred for carrying out such additional works would be borne by the defaulting mine operator.

20.7.1 Permitting Framework

French Guiana's mining regime is governed by the legislative and regulatory regime applicable to the French mainland with the exception of certain legal and regulatory provisions which are specific to it in order to take into account particular characteristics and constraints of this overseas territory. Information regarding the 2016 Mining Code reforms, and their potential effect on the permitting of the Project, is provided later in this section.

French Guiana developed a Departmental Mining Plan in 2011 which "defines the terms and conditions applicable to mining prospection [exploration], as well as the terms of the implementation and exploitation of land mining sites" with a view on economic sustainability as well as environmental protection. The general provisions of the Mining Code provide for two types of mining titles: the exclusive exploration permit ("*permis 243eochemic de 243eochemic*" or PER) for the exploration phase, and the concession (Concession) for the exploitation phase. A PER grants exclusive rights to carry out exploration activities within a specified exploration area. It is granted for an initial maximum period of five years, but can be renewed twice. A Concession confers on its holder an exclusive right, within the boundaries of such Concession, to explore and exploit the Mineral Resources that it covers. It is assignable and leasable, but cannot be mortgaged, and has an initial maximum term of 50 years and may be subject to successive 25-year renewal periods. Both the issuance of a PER and the granting of a Concession include public disclosure and participation in the permitting process.

In addition, small-scale mining, including most lawful alluvial operations, are carried out through exploitation authorizations ("*243eochemical243n d'exploitation*" or AEX) granted for areas no larger than 1 km^2. There are no current AEX operations within the Project area.

The Project is comprised of eightmining concessions covering approximately 135 km^2. The mining concessions, combined with appropriate permits, allow large-scale mine operations, and are valid until December 31, 2018, with potential renewal for a maximum of 25 years conditional upon a number of conditions, not the least of which is proving economic viability.

The Project encompasses the concessions C02/24; C03-46; PERs Tanon and Cigaline. The Project also includes a pending application for an exclusive exploitation permit ("*permis d'exploitation*" or PEX) covering an additional 14.4 km^2 outside of the concession areas. The PEX, combined with appropriate operating and environmental permits, also provides for medium- to large-scale mine operations, and is granted for five years with two potential and maximum renewals of five years each. The Project concessions, and the pending PEX, require quarterly reporting to the State, but carry no defined financial commitments for maintenance.

20.7.2 Facilities Classified for Environmental Protection (ICPE)

The French Environment Code has specific regulations for facilities owned or operated by any public or private natural or legal person, which may present dangers or inconveniences for neighbours, health, safety, public hygiene or the environment. These *Facilities Classified for Environmental Protection*, or ICPE, are subject to authorization, registration or declaration depending on the extent of the dangers or inconveniences caused by their operation. Included in these are:

- Ore processing-related infrastructure,
- Tailing storage facility and WRDs.
- Energy production infrastructure,
- Storage and fabrication of explosives, and
- Ancillary activities (e.g., hydrocarbon storage and distribution, hazardous goods storage, power generators, workshops, waste management, base camp, etc.).

A detailed breakdown and discussion of the various ICPE facilities and the classification thresholds is presented in the *Environmental Scoping Study* (WSP, 2015).

20.7.3 Restoration of the Access Road from the Croisée d'Apatou

The rehabilitation of the Montagne d'Or site access road from the Croisée d'Apatou will be subject to an environmental impact assessment and public enquiry, since these activities could lead to changes in the long- and cross- profiles of the minor beds of creeks crossed by the road, or the diversion of these creeks.

20.7.4 Law on Water and Aquatic Environments

Various activities necessary for the development of the Project will be subject to the Environment Code and its requirements, including:

- Development of process and potable water supplies;
- Stormwater which contacts mining facilities;
- TSF;
- Creek crossing structures;
- Diversion of natural drainages;
- Facilities located in designated flood zones;
- Process water ponds; and
- Mining infrastructure.

20.7.5 European Directives

Through its association as a Department of France, the Project will be subject to the European Directive 2010/75/EU on industrial emissions, or IED, which was established for environmental protection through the pollution prevention. Its guiding principles are:

- The use of BAT for the subject activities;
- The periodic review of the authorization conditions; and
- The restoration of the site to a state at least equivalent to that described in a "Baseline Report" which describes the state of the soil and groundwater prior to commissioning.

The activities covered by the IED's were introduced into the ICPEs. The directive also gives a list of criteria to be taken into consideration for determining the BATs. The BATs are compiled in reference documents (BREFs), which are produced by the European Commission's European Integrated Pollution Prevention and Control (IPPC) Bureau, or EIPPCB. The Project could be covered by the available BREFs on:

- Ferrous metals processing (December 2001 BREF); and/or
- The management of tailings and waste rock in mining activities (January 2009 BREF).

The IED also introduces the requirement to submit a Baseline Report describing the state of the soil and groundwater prior to commissioning of the mining Project. This report is to be used for reference purpose during final closure.

The Project will likely be covered by the ICPE Section 3250 – a) (production of non-ferrous crude metals from ore, concentrates or secondary raw materials by metallurgical, chemical or electrolytic processes). It will therefore be subject to the IED and require a Baseline Report, much of which has already been developed by GéoplusEnvironnement (WSP, 2015).

20.7.6 Mine Code Reformation

An original proposal and legislation for reformation of the French Mining Code, announced in 2012, failed to garner sufficient support for immediate passage. The proposal maintained much of the "French mining model" which is based on the ownership of the subsoil by the State (beneath 30 m) and the granting of permits for the exploration or exploitation of Mineral Resources. A revised version of the proposed code – departing from the draft announced in 2012 – was published for public comment on March 17, 2015. The primary reformations at that time included:

- Bringing the Mining Code into compliance with certain environmental principles;
- Providing legal certainty for carrying out mining activities (i.e., protection of mining operators legal position and tenures);
- Simplification of administrative procedures; and
- Inclusion and strengthening of public participation and transparency in the permitting process.

The draft legislation also proposed modifications to the current tax structure.

One of the major issues raised by operators with respect to the previous Mining Code was the general length of time it can take to assess, process, and grant mining titles. For example, under the previous framework, an application for a mining concession was deemed to be rejected if no explicit decision was taken by the minister within three years. A shortened period is provided for in the current version: decisions must be taken by the administration within six months (for the granting of exploitation/concession titles) or nine months (for the granting of exploitation titles), with the possibility to extend this time period once for a maximum of the same duration. A decree shall provide whether a lack of response from the administration at the end of this period amounts to refusal or approval of the application.

The previous Mining Code failed to take properly into account environmental concerns. Under the new code, the environmental impact of a proposed mining operation will be taken into account at the mining title stage, as opposed to construction and operation authorization stage. No longer will mining titles be granted without the administration having a clear understanding of the Project and potential impacts.

Finally, the new code reaffirms the principle that the entity in charge of exploration or the holder of the mining title can be held liable for damages caused by its activity. It provides that if the mining title holder is subject to insolvency proceedings, the court dealing with the proceedings may demand that the entity controlling the insolvent entity remedy any damages resulting from mining operations.

The Legislation to adapt the mining code to environmental law (n°4043) was filed on September 14, 2016, and was adopted in first reading by the National Assembly on September 21, 2016. Special provisions in the new Mine Code specific to French Guiana include, but are not limited to:

- The previously discussed zoning limitations for mining, taking into account the need to protect sensitive natural environments, landscapes, sites and populations and to manage in a balanced manner space and natural resources; and
- Consideration of the economic interest of French Guiana and the sustainable development of its Mineral Resources;

Proposed mining projects are subject to an environmental assessment pursuant to the Article L. 122-4 of the Environmental Code, and shall be made available to the public for two months. The public is advised of the consultation arrangements at least two weeks before it is made available.

The revised mine plan (amended based on comments and proposals collected during the initial review), is then forwarded for advice to the *Conseil Régional*, the *Conseil Général of Guyana*, the municipalities concerned, the Departmental Mine Commission, and the *Chambres Consulaires*. The proposal is deemed approved is no intervening action is taken by these entities after three months following submission. The plan is then finalized by the State representative in the Department and approved by decree by the *Conseil d'État*.

In areas where mining activities are prohibited, and in areas where it is prohibited except for underground operations and aerial exploration, the validity of exploration permits and operation permits valid at the time of its enactment can only be extended once. In the same areas, holders of an exclusive exploration permit can obtain an operation permit whose validity cannot be extended.

For substances other than liquid hydrocarbons or gas, the demand for exclusive exploration permits is not subject to competition if the requested area is below a threshold set by decree by the State Council.

20.7.7 Permitting Status and Schedule

As noted above, the initiation of permitting of the Project is dependent first and foremost on the completion of the mine plan being developed as part of this BFS. Typically, larger mining operations such as this have the benefit of a prefeasibility study (PFS) stage of analysis and development from which permitting is generally initiated. With the completion and publication of this BFS, the permitting of the Project can only now commence.

It is currently envisioned that the permitting process will require at least two years to complete for the mine, plant, and explosives emulsion plant. Several items have been initiated, are ongoing, or are currently planned (as noted below):

- Concession Montagne d'Or renewal application: *submitted December 2016*;

- Modification of the Local Urbanism Plan (PLU of SLM municipality: city council meeting on February 21, 2017 to validate PLU modification. Awaiting DEAL validation of modification and final publication. No specific date provided;
- Mining and ICPE permits: to be submitted Q3-2017. This will include:
  - Mine exploitation permit under Mining Code: open pit, topsoil stockpiles, haul road and access road (uncertain);
  - ICPE under Environmental Code: processing plant, fuel and hazardous storage, TSF, WRDs and base camp.
- Environmental permits that may have to be submitted: Road renovation, Power line construction; and
- Referral to French National Public Debates: to be submitted Q2-2017 following publication of NI43-101 Technical Report.

Each major permit application must include an EA which includes Avoid-Reduce-Compensate measures, and a specific focus on endangered species; a HS evaluating major risk scenarios for the Project define preventive and protective measures; as well as relevant technical studies supporting the findings of the EA and HS.

Nordgold will likely be required to submit a detailed Project description, predicated on the mine plan developed as part of this BFS, to the French National Public Debates (CNDP). This process can take from two to nine months to complete, depending on the review and recommendations for public debates made by the CNDP. Fortunately, however, the public debate process, and the Mining/ICPE permitting process, can run concurrently.

The currently envisioned draft permitting schedule, as proposed by Nordgold, is as follows:

- EA & HS new scoping study: March 2017;
- Official Scoping Decision from Prefect: May-June 2017;
- EA & HS final version submitted to Prefect/DEAL: September 2017; and
- Referral to CNDP and recommendation issuance: March, May 2017.

In the event that the CNDP recommendation is for national public debate, the process is not likely to be completed before the 2nd Quarter of 2018.

20.8 Reclamation and Closure

Upon final closure, the operator is required to provide an assessment of the final soil and groundwater conditions in comparison to the previously developed IED baseline report developed by Geoplus Environnement (2017). The operator is required to restore the site to a state that is, at a minimum, similar to that described in the baseline report (articles L. 515-30 and R. 515-75 of the Environment Code). This requirement is in addition to those regarding the restoration for the selected future land use (article L. 512-6-1 of the Environment Code). For new facilities, this report is part of the authorization request.

The objective of reclamation activities is to provide long-term stability, waste containment (to avoid both migration of pollutants and waste and minimize the risk of oxidation, leachate generation, and release of heavy metals), erosion prevention to reduce impact on the environment per the French Environment Code, Directive 2006/21/EC on the management of waste from extractive industries, and IED Directive concerning integrated pollution prevention and control. In order to demonstrate feasibility and permitability at this early stage of the Project, reclamation and closure of the earthworks facilities will be in accordance with the "Order of 15 February 2016 relating to non-hazardous waste storage facilities" and BAT Reference Document for the Management of Waste from the Extractive Industries (draft document, June 2016). Following the development of the ESIA, and associated environmental management plans, Nordgold may have an opportunity to modify these closure approaches during detailed design when more information has been developed, and equivalent levels of environmental protection can be effectively demonstrated.

Proposed reclamation and closure approaches and assumptions used to develop the closure cost estimate are summarized as follows:

- Most reclamation and closure activities will be carried out during the "active closure" period following the end of mining. The active closure period will last approximately five years, with most reclamation and closure activities completed in four years. The active closure period will be followed by a period of reclamation maintenance and monitoring activities that last approximately 30 years.
- WRDs will be reclaimed concurrently during operations. At the end of operations, some areas of the WRSAs may have potentially acid-generating materials exposed on the surface. In these areas, which will be minimized to the maximum extent possible using selective segregation and placement of materials (to be developed in the site Waste Rock Management Plan during the ESIA process), the closure covers will be constructed to meet regulatory and BAT guidance. For longer-term management of seepage from the WRSAs, passive "treatment" or polishing of the water could likely be possible with the use of constructed wetlands. Nominal costs for this activity have been included.
- For the TSF, post-closure supernatant water will be treated and discharged to the open pit following the hydraulic placement of a thin inert tailings cover to minimize the potential oxidation of sulphide materials during the tailings consolidation period. The tailings pipeline will then be removed and a cover will be mechanically placed which meets regulatory and BAT guidance. The tailings surface will be revegetated with a grass mix for stabilization, and surface water from the final engineered cover will be directed to the closure spillway for discharge to the environment.
- Safety berms will be constructed around the northern perimeter of the open pit. To enhance rapid infilling of the pit, the diversion ditches to the south of the pit will be backfilled and reclaimed during the closure period, and treated TSF supernatant water will be discharged to the pit. This action will aid in the inundation and submergence of exposed sulphide mineralization in the bottom of the pit, and reduce the potential for oxidation and ARDML in the longer term.
- The plant site will be decommissioned, decontaminated, and demolished. Building debris and wastes will be hauled to SLM or other site for final disposal. SRK has not assumed an on-site debris landfill for this activity. Remaining surface disturbances and yards associated with buildings will be regraded and covered with growth media, as necessary to achieve successful revegetation.
- According to Nordgold, ownership of the main powerline will be assumed by Électricité de France (EDF), Camp Citron and the airstrip will remain for future use, as will the primary access road. Project roads (i.e., interior access roads and haulage roads) will be reclaimed.

- Pond liners will be cut, excavated, and hauled off-site for proper disposal. The ponds will be regraded so as not to impound water, and revegetated. Once the haul road to the TSF is no longer required, the berm to the west of the CWP will be breached to allow for free flow.

The closure costs were calculated using the Standardized Reclamation Cost Estimator (SRCE) model, and are currently estimated at €51 million (US$56.1 million), based on an EURUSD of US$1.10:€1.00. With a contingency of 15%, the grand total is estimated at €58.7 million (US$64.6 million) at the same exchange rate.

20.9 International Standards and Guidelines

Even though French Guiana (as a Department of France) is a Designated Country with respect to the Equator Principles, Nordgold has committed to ensuring that Montagne d'Or is in compliance with international standards and guidelines, to the extent practicable, given the potential for international investment. Designated Countries are those countries deemed to have robust environmental and social governance, legislation systems, and institutional capacity designed to protect their people and the natural environment.

Potentially relevant international policies and/or guidelines for which the Project is likely to maintain compliance with include, but are not necessarily limited to:

- Equator Principles risk management framework for determining, assessing and managing environmental and social risk in projects;
- International Finance Corporation (Performance Standards) (IFC – PS) – social and environmental management planning;
- World Bank Guidelines (Operational Policies and Environmental Guidelines);
- Vienna Convention for the Protection of the Ozone Layer;
- Montreal Protocol on Substances that Deplete the Ozone Layer;
- Basel Convention on the Control of Trans-boundary Movements of Hazardous Wastes and their Disposal;
- Rotterdam Convention on the Prior Informed Consent Procedure for Certain Hazardous Chemicals and Pesticides in International Trade
- Stockholm Convention on Persistent Organic Pollutants;
- United Nations Climate Convention and the Kyoto Protocol; and
- Grenelle Environment Round Table of 2007 and the Grenelle Law II.

Table 20-1 provides a brief assessment of the approach to compliance anticipated for Montagne d'Or with respect to the IFC Performance Standards, even though the French Guiana is a Designated Country. As noted above, the fact that the overall permitting process has not yet been initiated means that compliance with many of these performance standards are pending.

Table 20-1: IFC Performance Standard vs. Compliance Approach

IFC Performance Standard (PS)	Summary of Requirements	Project Compliance
PS1: Assessment and Management of Environmental and Social Risks and Impacts	Development of an ESMS appropriate to the nature and scale of the Project which includes a policy, identification of risks and impacts, management programs, organizational capacity and competency, emergency preparedness and response, stakeholder engagement, monitoring and review.	Project will be subject to environmental impact assessment and environmental management requirements at various stages of the permitting process
PS2: Labour and Working Conditions	Identification of risks, impacts and management requirements associated with working conditions and terms of employment, non-discrimination and equal opportunity, retrenchment, grievance procedures, child labour, forced labour, occupational health and safety, third party workers and the supply chain.	Project will be governed by French and EU statutes and regulations, as well as local requirements
PS3: Resource Efficiency and Pollution Prevention	Promotes technically and financially feasible options to address resource efficiency (including greenhouse gas production and water consumption) and pollution prevention (with respect to wastes, hazardous materials management and pesticide use) across the Project life-cycle.	Project will be governed by French and EU statutes and regulations, as well as local requirements, and some international standards (e.g., WHO, etc.)
PS4: Community Health, Safety and Security	Evaluation of risks and impacts to the health and safety of Project-affected communities over the Project life cycle. Issues to be considered include infrastructure and equipment design and safety, hazardous materials management, ecosystem services, community exposure to disease, emergency preparedness and response, and management of security personnel.	Project will be governed by French and EU statutes and regulations, as well as local requirements, and some international standards (e.g., WHO, etc.)
PS5: Land Acquisition and Involuntary Resettlement	Applies to physical and or economic displacement resulting from Project acquisition of land rights or land use rights through expropriation, compulsory procedures, or negotiated settlements that if fail result in compulsory procedures. This PS also applies to Project situations requiring eviction of people occupying land without formal, traditional or recognizable usage rights and situations involving involuntary restrictions on land use or use of natural resources.	Due to its remoteness and location, there will be no involuntary resettlement associated with the Project.
PS6: Biodiversity Conservation and Sustainable Management of Living Natural Resources	Identification of risks and impacts on biodiversity and ecosystem services, especially focusing on habitat loss, degradation and fragmentation, invasive alien species, overexploitation, hydrological changes, nutrient loading and pollution. Guidance measures are dependent on type of habitat present (i.e. modified, natural or critical). Where a Project is likely to adversely impact ecosystem service, a systematic review to identify priority ecosystem services is required.	Project will seek to avoid impacts on biodiversity (critical habitat and species of interest for conservation) and ecosystem services. When avoidance of impacts is not possible, measures to minimize impacts and restore biodiversity and ecosystem services will be implemented and finally significant residual impacts will be offset.
PS7: Indigenous Peoples	Avoidance of adverse impacts on indigenous peoples and active engagement with the affected communities. Free, prior and informed consent (FPIC) of affected communities of indigenous peoples is required for projects with potential impacts to lands and natural resources subject to traditional ownership or customary use, relocation of indigenous peoples from such lands, and impacts to critical cultural heritage.	There are some Native American and *Bushinengues* communities in the region. The Project will avoid adverse impacts of projects on communities of indigenous peoples, or when avoidance is not possible, to minimize and/or compensate for such impacts. It will ensure full respect for the human rights and promote sustainable development benefits and opportunities for indigenous peoples.
PS8: Cultural Heritage	Promotes protection of cultural heritage in Project design and execution including implementation of chance find procedures, consultation, and community access and mitigation hierarchy. Critical cultural heritage should not be removed, significantly altered or damaged.	Operator will work with the Directorate of Cultural Affairs (DCA) and/or Regional Archaeology Department (SRA) to ensure that no cultural heritage is impacted by the Project; and that appropriate mitigation is employed in situations where cultural resources are encountered.

21 Capital and Operating Costs

21.1 Capital Cost Estimates

The overall Project Capital Cost Estimate (CCE), which is comprised of the initial and sustaining capital costs, was compiled from inputs developed by Lycopodium, SRK and Nordgold and reflects the Project scope as described in the study report. The estimate was based on an implementation strategy using a combination of Owner (self-perform) for the mining and earthworks for the treatment plant, infrastructure, roads, camp, TSF and water management dams, in addition to EPCM for all other plant, infrastructure, camp, TSF and water management scope.

An amount of contingency has been provided in the estimate to cover anticipated variances between the specific items allowed in the estimate and the final total installed Project cost. The contingency does not cover scope changes, design growth, etc. or the listed qualifications and exclusions. Contingency has been applied to the estimate as a deterministic assessment by assessing the level of confidence in each of the defining inputs to the item cost being engineering, estimate basis and vendor or contractor information. It should be noted that contingency is not a function of the specified estimate accuracy and should be measured against the Project total that includes contingency.

The CCEs for the Project were divided as follows:

- Lycopodium: Construction indirects, process plant (including water treatment and tails handling), reagents and plant services, and infrastructure capital costs (roads, power, offices, mine buildings, etc.), management costs, owner costs, and unit costs for the TSF and site water management; and
- SRK: Mining capital cost, and material quantity estimates for the TSF and site water management. SRK compiled the capital cost estimated for the TSF and site water management based on the unit costs provided by Lycopodium.

The estimate is subject to the following qualifications:

- Prices of materials and equipment with an imported content have been converted to an EURUSD of US$1.10:€1.00. All pricing received has been entered in its native currency;
- Concrete imported materials for construction have been included in the concrete installation rates by contractor;
- Construction contractor rates include mobile equipment, vehicles, fuel, construction power and consumables for the duration of construction. Potable water and raw water supply will be provided by Nordgold and will be available at site for the use by contractors;
- Mobilisation, demobilisation and rest & relaxation (R&R) flights of the construction contractor personnel are incorporated in the contractor indirect labour rates on the basis of individual contractors;
- Contractor accommodation costs per day have been included in the individual contractor's rates;
- Duties, taxes, working capital, capitalized interest, exchange rate fluctuations, escalation and land acquisition costs were excluded; and
- The CCE is considered to have an accuracy of ±15%, and is based on prices obtained during the third quarter of 2016 (3Q16) unless otherwise indicated.

21.1.1 Mining Capital Cost Estimates

Major mining capital costs were categorized into mining equipment and pre-production costs (both estimated by SRK), and mine support infrastructure (estimated by Lycopodium). Additional equipment units and replacement units purchased in Year 1 (2022) or later were calculated as sustaining mining equipment capital costs.

Capital cost estimates for major mining equipment (drills, loading equipment, haul trucks, dozers, graders, etc.) were based on quotes from equipment manufacturers (such as Atlas Copco, Komatsu and Caterpillar). Capital cost estimates for mining support equipment were based on quotes from Scania, or from the November 2016 Infomine mining cost reference guide (InfoMine, 2016).

The mining equipment initial capital cost estimate was based on the following:

- All mining units are based on new equipment purchases;
- Freight cost for some mining equipment was generally estimated at 8%, but varied according to quoted information available (and ranged between 3.7% and 12.0%);
- No import duties were deemed to be applicable;
- Allowances were made for on-site equipment erection costs for particular units;
- Mining equipment rebuilds (overhauls) were included in mining capital costs (but mostly in mining sustaining capital costs);
- The total mining equipment capital cost estimate did not include a contingency, since it is expected that equipment cost quotes can be reduced at the time of actual purchasing; and
- The mining equipment capital cost estimate noted in this section does not include the mining support infrastructure items.

Pre-Production Mining Operations

Pre-production mining costs will take place in Years -2 and -1 (2020, 2021). Pre-production mining tonnages were planned to be relatively low in order to reduce initial capital costs.

The pre-production mining cost includes cost allowances for clearing the initial mining operations area, establishing initial haul roads, initial WRD areas, low-grade ore stockpile area, and initial soil stockpile areas.

Mining Sustaining Capital Cost Estimate

Mining equipment rebuilds (overhauls) were included in the mining sustaining capital costs. These were estimated based on a total of 75% of the original cost of the equipment unit over the operating life of the machine, and scheduled as three overhauls during the operating life.

The LoM mining sustaining capital primarily consists of additional and replacement mining equipment. LoM mining capital costs, inclusive of contingency, are summarized in Table 21-1.

Table 21-1: Life-of-Mine Mining Capital Costs (000's)

Description	Euro €	US$ @ 1.10
Initial Capital Costs		
Preproduction Costs	49,797	54,843
Mining	66,444	73,088
Total Initial Capital	**€116,241**	**$127,931**
Sustaining Capital Costs		
Mining	59,085	$64,993
Total Sustaining Capital	**€59,085**	**$64,993**
Total Capital Cost	**€175,326**	**$192,924**

Source: SRK, 2017

21.1.2 Process/Infrastructure Capital Cost Estimate

Preliminary engineering drawings have been produced with sufficient detail to permit the assessment of the engineering quantities for earthworks, concrete, steelwork, mechanical and electrical for the crushing plant, processing plant, conveying systems and infrastructure.

LoM process and infrastructure capital costs, inclusive of contingency, are summarized below in Table 21-2. Infrastructure sustaining capital is restricted to stockpile FEL replacements every five years, annual commuter bus and light vehicle replacements, and a one-time WTP construction in Year 2.

Table 21-2: Initial TSF/Process/Infrastructure and Sustaining Process/Infrastructure Capital Costs (000's)

Description	Euro €	US$ @ 1.10
Initial Capital Costs		
TSF/Process/Infrastructure	376,088	413,697
Total Initial Capital	**€376,088**	**$413,697**
Sustaining Capital Costs		
Process	-	-
Infrastructure	12,536	13,790
Total Sustaining Capital	**€12,536**	**$13,790**
Total Capital Costs	**€388,624**	**$427,487**

Source: SRK, 2017

21.1.3 TSF Capital Cost Estimate

SRK developed the BFS CCE for the TSF that generally included the construction of the TSF proper (earthworks and geosynthetics). The tailings slurry discharge system and reclaim water system are included in the Process capital cost estimate developed by Lycopodium.

TSF Basis of Quantity Estimate

SRK developed a Basis of Estimate (BOQ) estimate for the TSF, which included a list of pay item, scope of work, basis of payment, quantity estimate and corresponding contingency for the quantities based on the confidence of the quantity estimate.

TSF Unit Rate Estimate

Lycopodium developed the unit rates and corresponding contingency for the TSF BOQ estimate developed by SRK.

TSF Capital Cost Estimate

TSF capital costs, inclusive of contingency, are summarized below in Table 21-3.

Table 21-3: Life-of-Mine TSF Capital Costs (000's)

Description	Euro €	US$ @ 1.10
Initial Capital Costs		
TSF*	26,938	29,632
Total Initial Capital	**€26,938**	**$29,632**
Sustaining Capital Costs		
TSF	144,177	158,595
Total Sustaining Capital	**€144,177**	**$158,595**
Total Capital Costs	**€171,115**	**$188,227**

*Included in Lycopodium's TSF/Process/Infrastructure Initial Capital Estimate in Table 21-2.
Source: SRK, 2017

21.1.4 Site Water Management Capital Cost Estimate

SRK developed the BFS capital cost estimate for the site water management infrastructure that generally included the infrastructure needed to divert non-contact water around the site and to collect, route, and store contact water. These costs include channel and sediment pond construction and pumping and conveyance systems, with the exception of surface water infrastructure associated with the TSF, the process and MSA. The surface water diversion structures associated with the TSF have been included in the TSF capital cost estimate developed by SRK, and the pumping and conveyance of tailings reclaim and raw water makeup have been included in the process capital cost estimate developed by Lycopodium.

Site Water Management Basis of Quantity Estimate

SRK developed a BOQ estimate for the site water management, which included a list of pay items, quantity estimate and corresponding contingency for the quantities based on the confidence of the quantity estimate.

TSF Unit Rate Estimate

Lycopodium developed the unit rates and corresponding contingency for the site water management BOQ estimate developed by SRK.

Site Water Management Capital Cost Estimate

LoM mine water and surface water management capital costs, inclusive of overall 12.3% contingency, are summarized below in Table 21-4.

Table 21-4: Life-of-Mine Water Management Capital Costs (000's)

Description	Euro €	US$ @ 1.10
Subtotal Initial Cost	**€9,769**	**$10,746**
Subtotal Sustaining Capital	**€5,030**	**$5,533**
Total Capital Costs	**€14,799**	**$16,279**

Source: SRK, 2017

21.1.5 Closure Costs

The closure/reclamation costs were calculated using the Standardized Reclamation Cost Estimator (SRCE) model, and are currently estimated at €51 million (US$56.1 million), based on an EURUSD of US$1.10:€1.00. With a contingency of 15%, the grand total is estimated at €58.7 million (US$64.6 million) at the same exchange rate.

LoM closure/reclamation costs, inclusive of contingency, are presented in Table 21-5.

Table 21-5: Life-of-Mine Capital Closure Costs (000's)

Description	Euro €	US$ @ 1.10
Total Closure/Reclamation Costs	**€58,732**	**$64,605**

Source: SRK, 2017

21.1.6 Capital Cost Summary

Based on an EURUSD of US$1.10:€1.00, total capital costs totaling US$860 million including final closure/reclamation costs are summarized in Table 21-6. Table 21-7 shows an approximate 9.5% overall contingency has been applied to capital items, which is appropriate for a feasibility-level of analysis.

Table 21-6: Life-of-Mine Capital Cost Summary (000's)

Description	Euro €	US$ @ 1.10
Initial Capital Costs		
Preproduction Costs	49,797	54,843
Mining	66,444	73,088
TSF/Process/Infrastructure	376,088	413,697
Water Management	9,769	10,746
Total Initial Capital	**€502,098**	**$552,374**
Sustaining Capital Costs		
Mining	59,085	64,993
Process	-	-
Infrastructure	12,536	13,790
TSF	144,177	158,595
Water Management	5,030	5,533
Total Sustaining Capital	**€220,828**	**$242,911**
Total Capital Costs		
Preproduction Costs	49,797	54,843
Mining	125,529	138,082
TSF/Process/Infrastructure	376,088	413,697
Infrastructure (Sustaining)	12,536	13,790
TSF (Sustaining)	144,177	158,595
Water Management	14,799	16,279
Subtotal Capital Costs	**€722,926**	**$795,285**
Closure/Reclamation	58,732	64,605
Total LoM Capital Costs	**€781,658**	**$859,890**

Source: SRK, 2017

Table 21-7: Life-of-Mine Capital Cost Contingency Rate

Description	Total Capital Costs (US$000's @ 1.10)	% Contingency
Preproduction Costs	54,843	-
Mining	138,082	-
TSF/Process/Infrastructure	413,697	10.8
Infrastructure (Sustaining)	13,790	-
TSF (Sustaining)	158,595	14.0
Water Management	16,279	12.3
Closure/Reclamation	64,605	15.0
Total LoM Contingency	**$859,890**	**9.5**

Source: SRK, 2017

21.2 Operating Cost Estimates

The LoM operating cost had been generated using the mine schedules and costs developed by SRK, the plant feed schedule developed by SRK, the processing costs developed by Lycopodium and the general and administration and water management costs developed by SRK.

The Operating costs for the Project were divided as follows:

- Lycopodium: Processing costs (including reagents, power, maintenance, labour, water treatment and tails/reclaim handling), and infrastructure (including personnel transport to/ from the site), and unit costs for the site water management. Lycopodium compiled the site water management operating costs; and
- SRK: Mining and Site General & Administration (G&A) costs and material quantity estimates for the site water management.

The estimate is subject to the following qualifications:

- Any impact of foreign exchange rate fluctuations;
- Any escalation from the date of the estimate;
- Any contingency allowance;
- Tailings storage costs, including future lifts and rehabilitation (considered/included in other sections of this study);
- Government monitoring/compliance costs (considered/included in other sections of this Study);
- Gold refining and bullion transport and in-transit security of gold from site (considered/included in other sections of this Study);
- Diesel prices for the Project of US$1.53 /L for the first two years of operation reducing to US$1.42 /L thereafter. The price reduction is due to reduced transport costs once the diesel depot in SLM is established; and.
- The operating cost estimate is considered to have an accuracy of ±15%, and is based on prices obtained during the third quarter of 2016 (3Q16) unless otherwise indicated.

21.2.1 Mining Operating Cost Estimate

Mine operating costs were developed by SRK. The basis of the operating costs is an owner operated mine. Pre-production development mining costs were not included as part of the mining operating costs (and were assigned to pre-production mining costs).

The Project mining production schedule has previously described the production mining years occurring from 2022 through 2031. These years have been described alternatively in this section for the production mining years as Year 1 (2022) through Year 10 (2031). Low-grade stockpile re-handling takes place mainly in Year 11 (2032) and Year 12 (2033).

SRK estimated the required mining equipment fleets, required production operating hours, and manpower to arrive at an estimate of the mining costs that the mining operations would incur. The mining costs were developed from first principles. The mining operating costs were developed in the following categories:

- Production drilling;
- Production blasting (explosives and blasting accessories, blasting contractor costs, etc.);

- Production loading;
- Production hauling;
- Other mine operations (dozing, grading, other road maintenance operations, etc.);
- Support equipment operations (equipment fueling and maintenance, pit lighting, etc.);
- Miscellaneous operations (geotechnical pit slope monitoring, fleet dispatch operations, etc.);
- Mine engineering (mine technical personnel operations, including technical consulting);
- Mine administration and supervision (mine and maintenance supervision, offices, etc.); and
- Freight (for equipment supplies and parts, excluding freight for fuel and explosives).

Pit dewatering operations (pumps and piping) and personnel camp costs are not included in the mining costs.

The mining operating cost estimates include the following parameters:

- Diesel fuel cost for mining of €1.24/L in pre-production, and after €1.14/L (delivered to site);
- Blasting required for 85% of in situ tonnage mined from the pit (15% free-digging saprolite);
  - o Blasting powder factor of 0.22 kg/t for waste and 0.24 kg/t for ore (kg explosives per tonne of rock);
  - o 100% use of bulk emulsion (blended) explosives for blasting;
- Bulk emulsion cost of €2,300/t (isotanks at site) during pre-production, and after €1,18/t (emulsion plant); and
- No contingency is included in the mining operating cost estimates.

Table 21-8 summarizes the mining operating costs by cost inputs based on a EURUSD of US$1.10:€1.00.

Table 21-8: Mining Operating Costs by Cost Inputs (000's)

Description	**Euro €**	**US$ @ 1.10**
Labour	271,355	298,491
Fuel	229,385	252,324
Explosives	98,918	108,810
Equipment Part & Supplies	113,963	125,359
Other	11,066	12,173
Subtotal LoM Mining Cost	**€724,687**	**$797,156**
Less Preproduction Cost	(49,797)	(54,777)
Total Mining Operating Cost	**€674,890**	**$742,380**

Source: SRK, 2017

Table 21-9 summarizes the annual mine operating costs on a cost per tonne mined (from the pits) basis, and summarizes cost per ore tonne mined in Euros. Ore mined refers to Proven and Probable Mineral Reserves.

Table 21-9: Annual Mining Operating Costs (000's and €/t Mined)

Mine Year	Total Mined (kt)	Total €/t Mined	Total €/t Ore Mined	Euro €	US$ @ 1.10
1	22,392	2.621	11.218	58,681	65,549
2	31,083	2.094	11.900	65,091	71,600
3	34,534	2.057	12.524	71,051	78,156
4	35,091	2.133	14.578	74,852	82,337
5	27,644	2.322	11.234	64,185	70,604
6	32,631	2.154	11.575	70,274	77,301
7	25,364	2.524	14.673	64,030	70,433
8	29,224	2.127	12.624	62,157	68,373
9	26,365	2.365	11.322	62,363	68,599
10	22,843	2.579	11.043	58,919	64,811
11	977	13.716	47.679	13,402	14,742
12	-	-	-	9,886	10,875
Total	**288,149**	**€2.342/t**	**€12.58/t ore**	**€674,890**	**$742,380**

Source: SRK, 2017
Years 11 and 12 are mainly low-grade stockpile re-handling years and thus there are no properly representative operating costs per tonne mined for these years.

21.2.2 Process Operating Cost Estimate

The processing operating cost estimate was developed using the parameters specified in the process design criteria and are based on an annual plant feed throughput of 4.6 Mt/y. The operating cost estimate presented in this section includes all direct costs to allow production of gold bullion at the Montagne d'Or plant site.

The processing operating cost battery limits were:

- Ore delivered to the RoM bin;
- Tailings discharge from the tails pipeline to the TSF;
- Gold bullion in plant gold room safe;
- Site buildings including MSA and camp;
- Raw water at pump suction from the CWP; and
- Decant return water at pump suction from the TSF.

The fixed and variable components of the operating costs have been estimated by assessing the extent to which each item in each of the cost centres is a fixed or variable cost. For example, most of the operating supplies are variable costs with direct dependence on throughput rate, while the labour cost can be considered fixed. Operating costs are based on the mine and processing schedules developed by SRK and reflected in this study.

Operating costs were estimated for the initial two years of operation and for years 3 onwards because the diesel prices are higher, increasing the consumables cost estimate. Table 21-10 presents the process cost based on a weighted cost over the LoM.

Table 21-10: Process Operating Costs (000's)

Description	Euro €	US$ @ 1.10
Operating Consumables	269,651	296,616
Processing Power	195,550	215,105
Infrastructure Power	36,838	40,522
Maintenance Materials	45,028	49,531
Laboratory Consumables	3,622	3,984
Processing Labour	51,938	57,132
Subtotal LoM Process Cost	**€602,628**	**$662,891**
Less Preproduction Cost	-	-
Total Process Operating Cost	**€602,628**	**$662,891**

Source: SRK, 2017

21.2.3 Site G&A Operating Cost Estimate

Site G&A or G&A costs represent a recurring annual operating cost to cover all owner's expenses during operations, and include people to manage the following:

- Site Management;
- Health and safety;
- Human resources;
- Supply chain management;
- Information services;
- Finance;
- Community and social responsibility ;
- Environmental and permitting; and
- Site services.

The G&A costs assumed that there will be three main operating areas: Site; the local office in SLM which will handle all operating logistical matters plus most of the administrative staff such as Finance and Supply Chain Management; and the regional office in the French Guiana capital of Cayenne which will used mainly for government relations activity. Site G&A costs were built up by a combination of first principles, supplier quotes, and allowances based on an EURUSD of US$1.10:€1.00.

The salary and respective burden (contribution sociale généralisée or CSG) information were supplied by Nordgold's JV partner Columbus who have extensive experience working in the region. The main labour assumptions are:

- 4 x 12 hour crew rotations working 7 days in/7 days out;
- Except for salaried or administrative workers, all workers will work day shift on one rotation and night shift the following rotation;
- All night shift workers will receive a 7.5% premium to base salary;
- All personnel are bused to and from the town SLM approximately 4 to 8 hour drive away, depending on condition of access road and highway;
- Expatriate staff will have a 20% premium in salary as part of their compensation package;
- Labour CSG percentages of base salaries are estimated to range from 16.27% for unskilled labour employees up to a maximum 45.55% for middle level technical staff/superintendents which then consequently decrease incrementally to 41.74% for upper level managers;

- Labour costs associated with bus drivers is included in Site Services department but all other busing operating and maintenance costs are captured in processing operating costs; and
- Labour costs associated with running the WTP 24/7 are included in the E&P department but all other WTP operating costs are included in the processing costs.

Table 21-11presents the annual summarized costs for each G&A department during Year 4 (2025) which is the year of peak production output for the Project.

Table 21-11: Site G&A Annual Operating Costs (000's)

Description	Euro €	US$ @ 1.10
Site Management		
Salaries and Wages	414	456
Materials, Supplies, Consumables	150	165
Rents/Premiums/Travel	1,338	1,471
Subtotal Site Management	**€1,902**	**$2,092**
Health and Safety		
Salaries and Wages	320	352
Materials, Supplies, Consumables, Training	141	155
Subtotal Health & Safety	**€461**	**$507**
Human Resources		
Salaries and Wages	320	352
Recruitment/Relocation	1,227	1,350
L&D Training Programs	182	200
Subtotal Human Resources	**€1,729**	**$1,902**
Supply Chain Management		
Salaries and Wages	485	533
Materials, Supplies, Consumables	9	10
Contract Services/Head Office Support	229	252
Subtotal SCM	**€723**	**$795**
Information Services		
Salaries and Wages	335	368
IT Equipment/Licenses	847	932
Subtotal Information Services	**€1,182**	**$1,301**
Finance		
Salaries and Wages	522	574
Contract Services/Head Office Support	182	200
Subtotal Finance	**€704**	**$774**
Community and Social Responsibility		
Salaries and Wages	174	191
Materials, Supplies, Consumables	45	50
Community Funding	273	300
Subtotal CSR	**€492**	**$541**
Environmental and Permitting		
Salaries and Wages	634	697
Operating Costs	273	300
Contract Services	455	500
Subtotal E&P	**€1,361**	**$1,497**
Site Services		
Salaries and Wages	2,045	2,250
Contract Services	7,689	8,458
Site Facilities Maintenance	167	183
Subtotal Site Services	**€9,901**	**$10,891**
Grand Total	**€18,454**	**$20,299**

Source: SRK, 2017

Total Site G&A operating costs are summarized in Table 21-12.

Table 21-12: Site G&A Operating Costs (000's)

Description	Euro €	US$ @ 1.10
Site Management	22,820	25,102
Health & Safety	5,527	6,079
Human Resources	20,745	22,819
Supply Chain Management	8,670	9,537
Information Services	14,189	15,608
Finance	8,446	9,290
Community and Social Responsibility	5,906	6,497
Environmental and Permitting	16,331	17,964
Site Services	111,939	123,133
Subtotal LoM Site G&A Cost	**€214,572**	**$236,029**
Less Preproduction Cost	-	-
Total Site G&A Operating Cost	**€214,572**	**$236,029**

Source: SRK, 2017

21.2.4 Site Water Management Operating Cost Estimate

SRK developed the BFS operating cost estimate for the site water management that generally included the power costs to convey contact water from various locations around the site to the CWP or, later in the mine life to the pit Lake. Operating costs including electric power consumption or diesel fuel usage based on median annual flow rates, pumping heads, and typical pump and motor efficiencies. Electric power and diesel fuel costs were provided by Lycopodium. Operating costs associated with maintenance and operation of the mine water management system were addressed in the Site Services component of the G&A Operating cost estimate while labour costs are included in the Environmental and Permitting group of the G&A Operating cost estimate.

Total site water management operating costs are summarized in Table 21-13.

Table 21-13: Site Water Management Operating Costs (000's)

Description	Euro €	US$ @ 1.10
Electrical Power	2,965	3,262
Diesel Fuel	3,160	3,475
Subtotal LoM Site Water Management Cost	**€6,125**	**$6,737**
Less Preproduction Cost	(60)	(66)
Total Site Water Management Operating Cost	**€6,065**	**$6,671**

Source: SRK, 2017

21.2.5 Operating Cost Summary

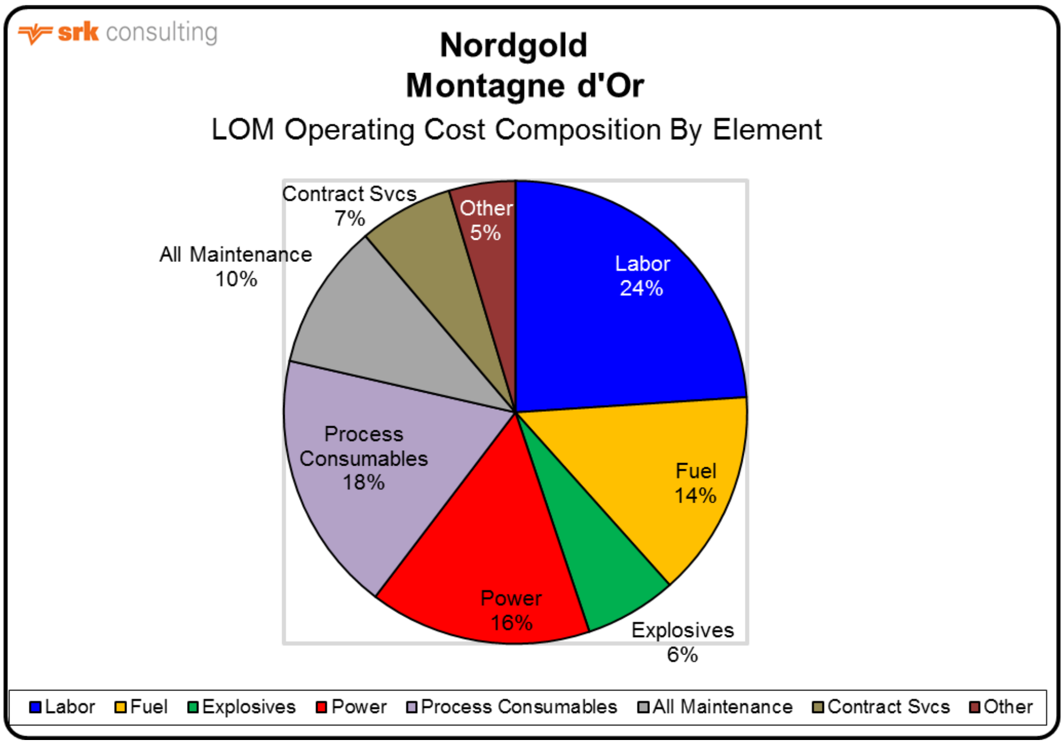
Based on an EURUSD of US$1.10:€1.00, Table 21-14 presents estimated total operating costs of US$30.45/t processed and Source: SRK, 2017

Figure 21-1 shows the relative composition of operating cost inputs such as labour, process consumables and power which are the three largest cost items at 24%, 18%, and 16%, respectively.

Table 21-14: Operating Cost Summary (000's and per Tonne Processed)

Description	Euro €	US$ @ 1.10
Mining	674,890	742,380
Process	602,628	662,891
Site G&A	214,572	236,029
Water Management	6,065	6,671
Total Operating Costs	**€1,498,155**	**$1,647,971**
Operating Cost Unit Rates	**€/t Proc.**	**US$/t Proc.**
Mining ($/t mined)	2.28	2.58
Mining ($/t processed)	12.47	13.72
Process	11.14	12.35
Site G&A	3.97	4.36
Water Management	0.11	0.12
Total Operating Costs	**€27.69**	**$30.45**

Source: SRK, 2017



Source: SRK, 2017

Figure 21-1: Operating Cost Composition

22 Economic Analysis

22.1 Principal Assumptions and Input Parameters

The indicative economic results summarized in this section are based upon work performed by SRK, Lycopodium or received from Nordgold in 2016. They have been prepared on both a combined monthly/quarterly and annual pre-tax and after-tax basis, a 100% equity basis with no Project financing inputs, and are in Q4 2016 U.S. constant dollars. However, the metrics reported in this volume are based on the annual cash flow model results.

Key criteria used in the analysis are discussed in detail throughout this section. Principal Project assumptions used are shown summarized in Table 22-1.

Table 22-1: Basic Model Parameters

Description	Value
Construction Start Year	2019
Commercial Production Year	2022
Construction Period	3 years
Open Pit Mine Life	11 years
Process Plant Life	12 years
Mine Operating Days per Year	365
Mill Operating Days per Year	365
Discount Rate	EOP @ 5%
EURUSD Base Rate For Cost Estimates	1.10
EURUSD Final Rate For Economic Results	1.05

Source: SRK, 2017

2019 was selected as the construction start year which reflects an assumption that a positive result from a BFS and an investment decision to proceed with the Project and financing arrangements will be made by the end of 2018. All costs incurred to that point are considered sunk with respect to this analysis.

The selected Project discount rate is 5% as directed by Nordgold and the valuation uses end-of-period discounting. A sensitivity analysis of the discount rate is discussed later in this section.

For this Project, the selection of an appropriate EURUSD is critical in the Project valuation. For example, the quotation EURUSD 1.1000 means that one euro is exchanged for 1.1000 US dollars. If the quote changes from EURUSD 1.1000 to 1.1500, the Euro has increased in relative value, because either the dollar buying strength has weakened or the Euro has strengthened, or both. On the other hand, if the EURUSD quote changes from 1.1000 to 1.0500 the Euro has become relatively weaker than the US Dollar.

For the study, a Base EURUSD of US$1.10:€1.00 was originally selected in mid-2016 as the basis for cost estimates and economic results. However, in early Q1, 2017, Nordgold requested the economic results to reflect a EURUSD of US$1.05:€1.00 based on recent long term trend forecasts. It was decided at that time not to change the cost estimates and therefore, the source capital and operating cost inputs to the TEM still reflect the 1.10 exchange rate. However, a conversion mechanism to convert costs to USD @ 1.05 exchange rate was developed for the cash flow calculation module of the TEM. The methodology to convert the original cost inputs to USD @ 1.05 exchange rate is as follows:

- Euro-Based Costs estimated at 1.10 EURUSD:
  Cost in Euros X weighted average % EUR basis X 1.05 exchange rate, plus
  Cost in Euros X weighted average % USD basis X ((1.05 exchange rate/1.10 Base Exchange rate) X 1.05 exchange rate)
  = Adjusted USD Costs @ 1.05 EURUSD
- USD-based Costs estimated at 1.10 EURUSD:
  Cost in USD x weighted average % Euro basis x (1.05 exchange rate/1.10 Base Exchange rate), plus
  Cost in USD x weighted average % USD basis.
  = Adjusted USD Costs @ 1.05 EURUSD.

The conversion from original capital and operating costs estimated at 1.10 rate to their equivalent values at 1.05 rate is shown in various tables in this section.

22.2 Cashflow Forecasts and Annual Production Forecasts

22.2.1 Mine Production

Table 22-2 is a summary of the estimated mine production over an 11-year mine life for the open pit operations. Ore mined refers to Proven and Probable Mineral Reserves. A full LoM annual cash flow forecast is presented in Appendix B.

Table 22-2: Life-of-Mine Production Summary

Description	Value
Ore Mined (kt)	54,114
Waste Mined (kt)	241,753
Total Material Mined (kt)	295,867
Strip Ratio	4.5
Mining Rate (kt/y)	35,091
RoM Grade (g/t)	1.58
Contained Gold (koz)	2,745

Source: SRK, 2017

22.2.2 Mill Production

A summary of the estimated process plant production for the Project is contained in Table 22-3 for a 12 year operating life. Ore processed refers to Proven and Probable Mineral Reserves. A full LoM annual cash flow forecast is presented in Appendix B.

Table 22-3: Life-of-Mine Process Production Summary

Description	Value
Total Ore Processed (kt)	54,115
Processing Rate (kt/y)	4,575
Processed Grade (g/t)	1.58
Contained Gold (koz)	2,745
Gold Recovery (%)	93.8%
Recovered Gold (koz)	2,574

Source: SRK, 2017

22.2.3 Revenue

Gold pricing assumptions used in the economic analysis include a constant base case (aka Neutral Case) LoM gold price of US$1,250/troy oz. In SRK's opinion, this price is appropriate, especially when contrasted against various benchmarks below:

- February, 2017 end of month closing price of US$1,257/troy oz;
- February, 2017 three year trailing average of monthly closing prices of US$1,215/troy oz; and
- Doré refining/selling costs used in the economic analysis are as follows:
  - 99.5% payable to European refinery customer; and
  - US$1/troy oz selling/refining plus transportation/insurance costs.

22.2.4 Capital and Operating Costs

Total capital costs totaling US$827 million including final closure/reclamation costs are summarized in Table 22-4 which also shows the translation of capital costs estimated at the initial EURUSD of US$1.10:€1.00 to the final US$1.05:€1.00 selected for the economic analysis. Approximately 9.5% overall contingency has been applied to capital items as shown in Table 22-5, which is appropriate for a BFS level of analysis in SRK opinion. Therefore, the total initial capital required to construct a 4.6 Mt/y project that will produce approximately 237 koz/y during the first 10 years of the operation is estimated to be US$535.2 million which includes US$52 million of preproduction costs.

Table 22-4: Life-of-Mine Capital Costs (000's)

Description	Euro €	US$ @ 1.10	US$ @ 1.05
Initial Capital Costs			
Preproduction Costs	49,797	54,843	52,003
Mining	66,444	73,088	69,047
TSF/Process/Infrastructure	376,088	413,697	403,991
Water Management	9,769	10,746	10,150
Total Initial Capital	**€502,098**	**$552,374**	**$535,191**
Sustaining Capital Costs			
Mining	59,085	64,993	61,208
Process	-	-	-
Infrastructure	12,536	13,790	13,477
TSF	144,177	158,595	151,282
Water Management	5,030	5,533	5,154
Total Sustaining Capital	**€220,828**	**$242,911**	**$231,120**
Total Capital Costs			
Preproduction Costs	49,797	54,843	52,003
Mining	125,529	138,082	130,255
TSF/Process/Infrastructure	376,088	413,697	403,991
Infrastructure (Sustaining)	12,536	13,790	13,477
TSF (Sustaining)	144,177	158,595	151,282
Water Management	14,799	16,279	15,304
Subtotal Capital Costs	**€722,926**	**$795,285**	**$766,312**
Closure/Reclamation	58,732	64,605	60,659
Total LoM Capital Costs	**€781,658**	**$859,890**	**$826,971**

Source: SRK, 2017

Table 22-5: Life-of-Mine Capital Cost Contingency Rate (000's)

Description	Total Capital Costs (US$ @ 1.10)	% Contingency
Preproduction Costs	54,843	-
Mining	138,082	-
TSF/Process/Infrastructure	413,697	10.8
Infrastructure (Sustaining)	13,790	-
TSF (Sustaining)	158,595	14.0
Water Management	16,279	12.3
Closure/Reclamation	64,605	15.0
Total LoM Contingency	**$859,890**	**9.5**

Source: SRK, 2017

An estimate of US$6.2 million of working capital was calculated for the first year of commercial production as shown in Table 22-6 using the EURUSD of US$1.05:€1.00. All working capital is recaptured by the end of the mine life with a LoM net free cash flow (FCF) impact of US$0.

Table 22-6: Working Capital Estimate in Year 1 (US$000's)

Description	1st Month	Full Year
Accounts Receivable	2,086	4,638
Accounts Payable	(7,249)	(8,086)
Opening Stocks – Mining	3,907	3,907
Opening Stocks – Processing	4,535	5,791
Total Working Capital	**$3,278**	**$6,249**

Source: SRK, 2017

The total operating cost unit rate of US$28.76/t processed is summarized in Table 22-7 which also shows the translation of operating costs estimated at the initial EURUSD of US$1.10:€1.00 to the final US$1.05:€1.00 selected for the economic analysis.

Table 22-7: Operating Cost Summary

Operating Costs in 000's	Euro €	US$ @ 1.10	US$ @ 1.05
Mining	674,890	742,380	704,040
Process	602,628	662,891	621,830
Site G&A	214,572	236,029	224,309
Water Management	6,065	6,671	6,368
Total Operating Costs	**€1,498,155**	**$1,647,971**	**$1,556,547**
Operating Cost Unit Rates	**€/t Proc.**	**US$/t Proc.**	**US$/t Proc.**
Mining ($/t mined)	2.28	2.58	2.44
Mining ($/t processed)	12.47	13.72	13.01
Process	11.14	12.25	11.49
Site G&A	3.97	4.36	4.15
Water Management	0.11	0.12	0.12
Total Operating Costs	**€27.69**	**$30.45**	**$28.76**

Source: SRK, 2017

22.3 Technical Economics

22.3.1 Economic Results

The TEM was prepared on an annual after-tax basis, the results of which are presented in this section. Key criteria used in the analysis are discussed in detail in Section 21.

Economic results are summarized in Table 22-8 and a full LoM annual cash flow forecast is presented in Appendix B.

The results indicate that at a US$1,250/oz gold price the Project returns an after-tax NPV 5% of US$370 million and IRR of 18.7%. Initial capital is estimated at US$535 million before US$174 million in surplus initial capital tax credit refunds (US$361 million net initial capital), sustaining capital at US$231 million, and a closure/reclamation capital cost estimated at US$61 million.

Table 22-8: Life-of-Mine Tax Indicative Economic Results (US$000's)

Description	Value
Market Prices	
Gold (US$/oz)	$1,250
Payable Metal	
Payable Gold (koz)	2,572
Total Gross Revenue	**$3,214,654**
Operating Costs	
Mining	(704,040)
Processing	(621,830)
Site G&A	(224,309)
Water Management	(6,368)
Selling/Refining	(2,375)
Royalties	(153,374)
Total Operating Costs	**($1,712,296)**
Operating Margin (EBITDA)	$1,502,358
Taxes	
Income Tax (Net of Tax Credits)	(200,746)
Total Taxes	**($200,746)**
Working Capital	(0)
Operating Cash Flow	**$1,301,612**
Capital	
Initial Capital	(535,191)
Sustaining Capital	(231,120)
Closure/Reclamation Capital	(60,659)
Total Capital	**($826,971)**
Surplus Tax Credit Refunds	**185,632**
Metrics	
Pre-tax Free Cash Flow	861,019
After-tax NPV @ 5%	**506,731**
Pre-tax IRR	**22.2%**
Pre-tax Undiscounted Payback from Start of Comm. Prod. (Years)	3.7
After-tax Free Cash Flow	660,273
After-tax NPV @ 5%	**369,949**
After-tax IRR	**18.7%**
After-tax Undiscounted Payback from Start of Comm. Prod. (Years)	4.1
All-In Sustaining Costs (AISC - $/oz)	$779

Source: SRK, 2017

22.3.2 All-in Sustaining Cash Costs

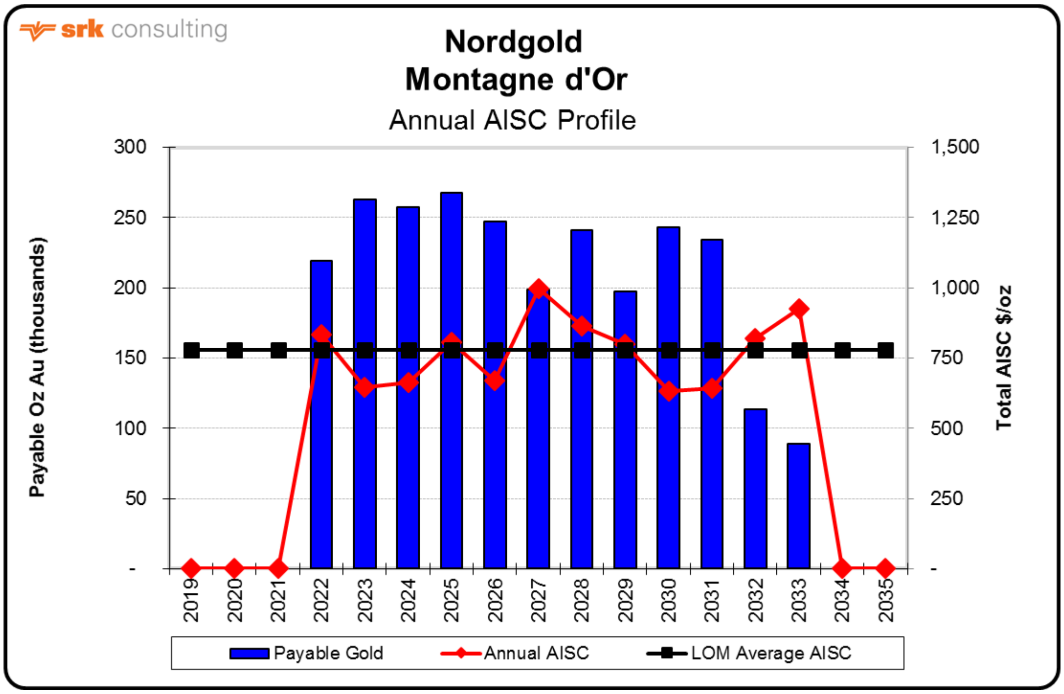
Table 22-9 shows the build-up of an LoM AISC of US$779/payable oz over the 12-year life of the Project which contains US$24/payable oz of final closure/reclamation costs which are incurred close the end of the mine life. Average LoM AISC before closure/reclamation capital is US$756/payable oz.

Table 22-9: Life-of-Mine All-In Sustaining Cost (AISC) Contribution

LoM Payable Gold (koz)		**2,572**
Description	**US$000's**	**US$/oz**
Mining	704,040	274
Processing	621,830	242
Site G&A	224,309	87
Water Management	6,368	2
Selling/Refining	2,375	1
Direct Cash Costs	**$1,558,922**	**$606**
Royalties	153,374	60
Indirect Cash Costs	**$153,374**	**$60**
Adjusted Operating Costs/Total Cash Costs	**$1,712,296**	**$666**
Sustaining Capital	231,120	90
Closure/Reclamation Operating/Capital	60,659	24
Corporate G&A	-	-
Off-Mine Exploration	-	-
Sustaining Costs	**291,780**	**$113**
Total LoM All-in Sustaining Costs	**$2,004,076**	**$779**
Total All-in Sustaining Costs w/o Final Closure	**$1,943,416**	**$756**

Source: SRK, 2017

Figure 22-1 shows the AISC trend during the mine operations against an overall average LoM AISC of US$779/payable oz over the 12-year life of mine at an average annual production rate of 214,000 oz Au per year and an average head grade of 1.58 g/t Au. The AISC variations are mainly driven by annual gold production and can range from US$633 to US$998 per oz in a given year. It should also be noted that during the first ten years of the operation the average annual production is 237,000 oz Au per year with an AISC of US$749/payable ounce and an average head grade of 1.73 g/t Au.



Source: SRK, 2017

Figure 22-1: Annual AISC Curve Profile

22.4 Taxes, Royalties and Other Interests

Changes to the French Tax Code which covers the mining taxation regime applicable to French Guiana are under discussion in the French Parliament at the time of writing this report. Due to this uncertainty, KPMG's French affiliate, FIDAL, was contracted to provide best efforts guidance on material tax issues affecting the Project. Thus, royalties and income taxes have been calculated for the Project with best efforts assumptions provided by SRK, Nordgold and FIDAL.

22.4.1 Royalties

The legal State royalty rates applicable on gold production in French Guiana for the year 2016 area are as follows and currently net to US$27.35/recovered oz:

- Municipal Royalty @ €137.90/kg Au (article 1519 of French tax code);
- Departmental Royalty @ € 27.50/kg Au (article 1587 of French tax code); and
- French Guiana tax @ € 672.01/kg Au (article 1599 quinquies B of French tax code).

In addition to State royalties, the Project has two additional royalty agreements with the following companies:

- Euro Ressources' NSR royalty of 1.8% up to 2 Moz, and 0.9% after 2 Moz and currently net to US$18.98/ payable oz;
- Sandstorm Resources' 1% NSR royalty which currently nets to US$11.99/ payable oz; and
- Overall effective NSR royalty rate is estimated to be 5.0% up to 2 Moz and 4.1% afterwards until the end of production.

22.4.2 Corporate Income Taxes (CIT)

The information in this section has been provided in most part by FIDAL (FIDAL, 2017) and are based on the standard French CIT regime regardless of the potential tax advantages that can be granted within the framework of the concession/specific agreement concluded by the State with Nordgold/Columbus. The taxation model is presented in Appendix C but the assumptions are listed as follows:

- French national income tax rate, currently 33.5%, will be reduced to 28% rate in Year 2020 and would be applied regardless of the level of taxation profit or the turnover;
- In addition, a 3.3% surcharge is currently in force and applies on the CIT portion exceeding €763,000 (US$801,150) and is not envisaged to be eliminated in the next coming years;
- Net Operating Losses (NOL) can be carried forward with no time limit but the amount of tax losses that may be carried forward is limited to €1 million (US$1.05 million) plus 50% of the tax profit exceeding €1 million of a given financial year;
- Columbus' carried forward losses of €8.6 million (US$9 million) through Sept, 2016 can be used up as soon as the first year of production;
- The depreciable base for acquired fixed assets take into account the actual purchase price as well as accessory costs such as transportation, customs duties, installation and assembly;
- French tax law includes a strict definition of permissible depreciation. Straight-line depreciation normally is used but for certain types of business property, including machinery and equipment used for manufacturing, processing, and transport associated with industrial buildings having a useful life of less than 15 years, French tax law allows for accelerated declining-balance (DB) depreciation for both fixed assets as well as moveable equipment such as trucks and shovels;
- SRK used a 5-year DB method with 1.75 multiplier for moveable equipment that begins in the year of acquisition and a 15-year DB method with 2.25 multiplier for major fixed assets such as the process plant (excluding cost of buildings) starting when the plant is placed into service;
- Both DB methods utilize switching to SL depreciation halfway through the asset life to maximize depreciation deductions along with an EoM write-off of all remaining depreciation during the last year of production;
- DB-eligible assets account for 84% of the depreciable asset base and the remaining 16% of capital assets are depreciated using 5-year straight-line method for equipment/machinery (rebuilds and other equipment) starting in the year acquired and 20 year straight-line depreciation for building-related assets (camp and auxiliary facilities) starting when placed into service. As with DB depreciation, there is an EoM write-off of all remaining depreciation during the last year of production;
- Final closure/reclamation capital were not considered to be part of the depreciable asset base;
- A tax credit is provided for in articles 244 quarter W, 199 ter U and 220Z of the French General Tax Code. It allows companies exceeding the turnover's threshold of €20 million to benefit from a tax credit against the corporate tax liability of the company with the excess of the credit being immediately refundable and is considered not a taxable item;

- This regime applies to new productive investments made in overseas departments such as French Guiana. The tax credit applies to initial investments only and to the assets which are new, depreciable and which will be used for business purposes during at least 5 years;
- The tax credit is set off against the corporate tax liability of the taxpayer, based upon its taxable income and does not impact the historical book and tax value of the assets which is used for depreciation purposes;
- At the time of writing this report, the rate of the tax credit is equal to 35% of eligible capital costs and in order to benefit from the credit, the qualifying investments must be made before the end of 2020 which corresponds to the end of the second year of construction for the Project;
- Nordgold/Columbus staff has been advised by various stakeholders that since the tax credit program was designed to help investment and economic development of unprivileged overseas territories and this goal had not been accomplished in French Guiana, there is little doubt that the tax credit program will be renewed. The base case for the Project evaluation therefore assumes that the tax credit program will continue through the LoM of the Project which ends in 2033; and
- Nordgold/Columbus staff, with the assistance of French financial consultants (Starinvest) familiar with the tax credit regulations, determined eligible capital costs (including all associated costs like freight, labour, etc. as well as contingency) with the methodology summarized below:
    - Moveable Assets (Initial and Sustaining Capital)
        - All major mobile mining equipment; and
        - All other equipment (but excluding initial spare parts).
    - Buildings Assets (Initial Capital Costs only)
        - Construction Indirect costs (but excluding cost of fencing, first aid facility, surveying and repairs/maintenance associated with Project construction phase);
        - All Treatment Plant costs;
        - All Reagents & Plant Services costs;
        - Infrastructure costs (but excluding site fencing and auxiliary facilities such as Administration/Security, Plant Offices, and all accommodation buildings);
        - All mining-related facilities costs (workshop, wash down facility, etc.);
        - All management costs (EPCM and vendors) related to constructing the Projects; and
        - Owners Project Costs such as plant mobile equipment and training (but excluding labour-related costs).
    - Other Assets (Initial and Sustaining Capital Costs)
        - Preproduction costs related to mining (but excluding maintenance parts, explosives, diesel, and electric power costs);
        - All Tailings Storage Facility earthworks costs during LoM operations;
        - All mine water management costs; and
        - All surface water management costs (excluding cost of replacement pumping equipment).

French national taxation policies materially affect the Project metrics with respect to corporate income tax and especially the French Overseas Department Tax Credit Program. With respect to the latter, from a total of US$680.2 million of eligible capital expenditures, US$238 million in tax credits was generated for the project based on the assumptions listed in this section, of which US$174.4 million were generated from eligible initial capital expenditures during the period of 2019-2021.

Due to timing considerations of these tax credits, approximately US$114.7 million of the total credits are eligible for direct refund in preproduction years 2019-2021 while the remaining US$59.7 million preproduction tax credit is eligible for direct refund in 2022. In addition, US$11.3 million in tax credits generated from eligible sustaining capital expenditures in 2022 are eligible for direct refund in 2023 bringing the total surplus tax credit refund receivable for the project during the period 2020-2023 to US$185.6 million. It should be noted that US$92,000 in surplus tax credit refunds were generated in the final years of the operation (2032-2034) but were not included in the project metrics since they are immaterial and likely not to be refunded by the government.

The remaining US$52.4 million of tax credits generated by eligible sustaining capital expenditures are used to lower tax annual payable amounts for the remaining life of the project without any direct refunds. The corporate income tax, tax credit, and depreciation calculations are presented in Appendix C and a sensitivity analysis of these critical cash flow items is discussed in the following section.

22.5 Sensitivity Analysis

Sensitivity analyses for key economic parameters are shown in Table 22-10 and Table 22-11. The Project is nominally most sensitive to gold grade and the EURUSD. The Project's sensitivities to capital and operating costs are similar but slightly more susceptible to operating costs.

Table 22-10: Sensitivity Analysis of After-Tax NPV 5%

NPV@5% (US$ Millions)	-20%	-15%	-10%	-5%	Base	5%	10%	15%	20%
Gold Grade	51	134	213	293	370	447	523	599	675
Operating Costs	530	491	451	411	370	329	288	246	204
Capital Costs	472	446	421	395	370	345	319	294	268
EURUSD	619	558	497	434	370	304	235	164	91

Source: SRK, 2017

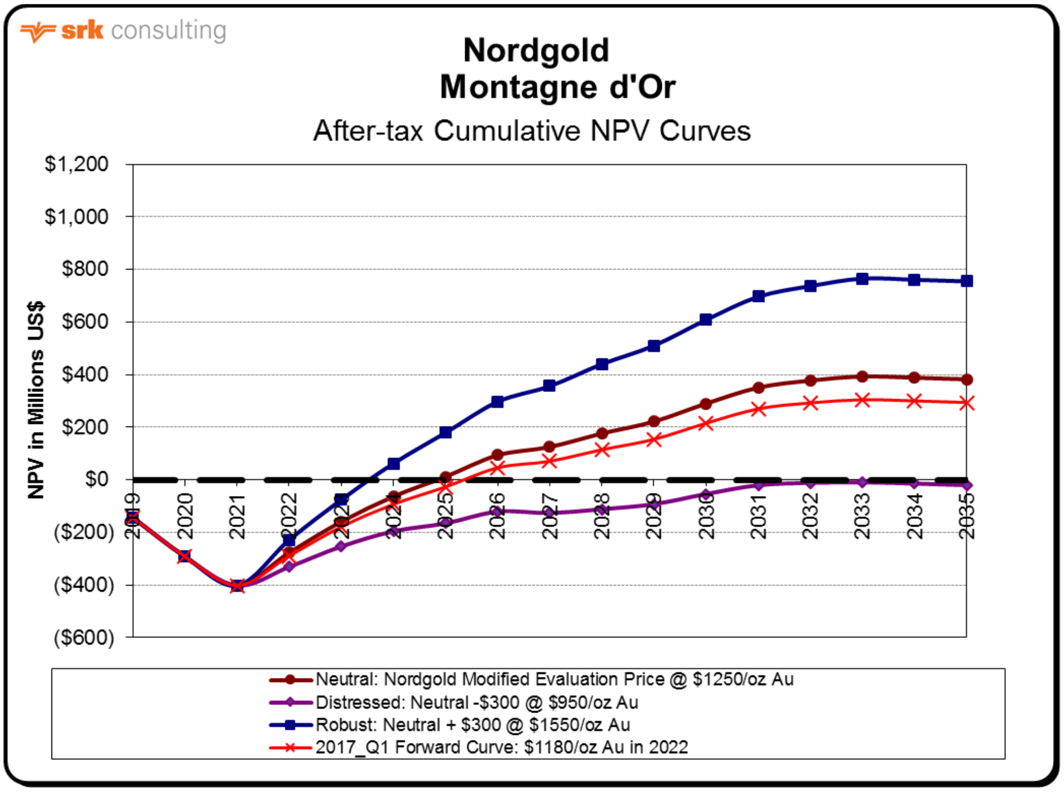
Table 22-11: Sensitivity Analysis of After-Tax IRR

IRR	-20%	-15%	-10%	-5%	Base	5%	10%	15%	20%
Gold Grade	7.4%	10.9%	13.8%	16.3%	18.7%	20.8%	22.8%	24.7%	26.5%
Operating Costs	23.2%	22.1%	21.0%	19.8%	18.7%	17.4%	16.2%	14.8%	13.4%
Capital Costs	26.0%	23.9%	22.0%	20.2%	18.7%	17.2%	15.9%	14.7%	13.5%
EURUSD	27.3%	25.2%	23.0%	20.9%	18.7%	16.4%	14.0%	11.5%	8.8%

Source: SRK, 2017

22.5.1 Gold Price Sensitivity

Additional gold price sensitivity analyses are shown in Figure 22-2 with after-tax Project NPV 5% at constant "Robust" prices (US$1,250/oz neutral price + US$300/oz = US$1,550/oz), and a constant "Distressed" prices (US$1,250/oz neutral gold price – US$300/oz = US$950/oz). Furthermore, SRK incorporated a forward price curve sensitivity using Consensus Economics' "Consensus Market Forecast" (CMF Forward Curve), which shows US$1,180/oz in 2022. All told, the after-tax Project NPV 5% changes approximately US$1.24 million for every US$1 change in gold price, either upwards or downwards. In addition, Table 22-12 also shows price sensitivity at a series of discrete price points.



Source: SRK, 2017

Figure 22-2: Gold Price Sensitivity Analysis

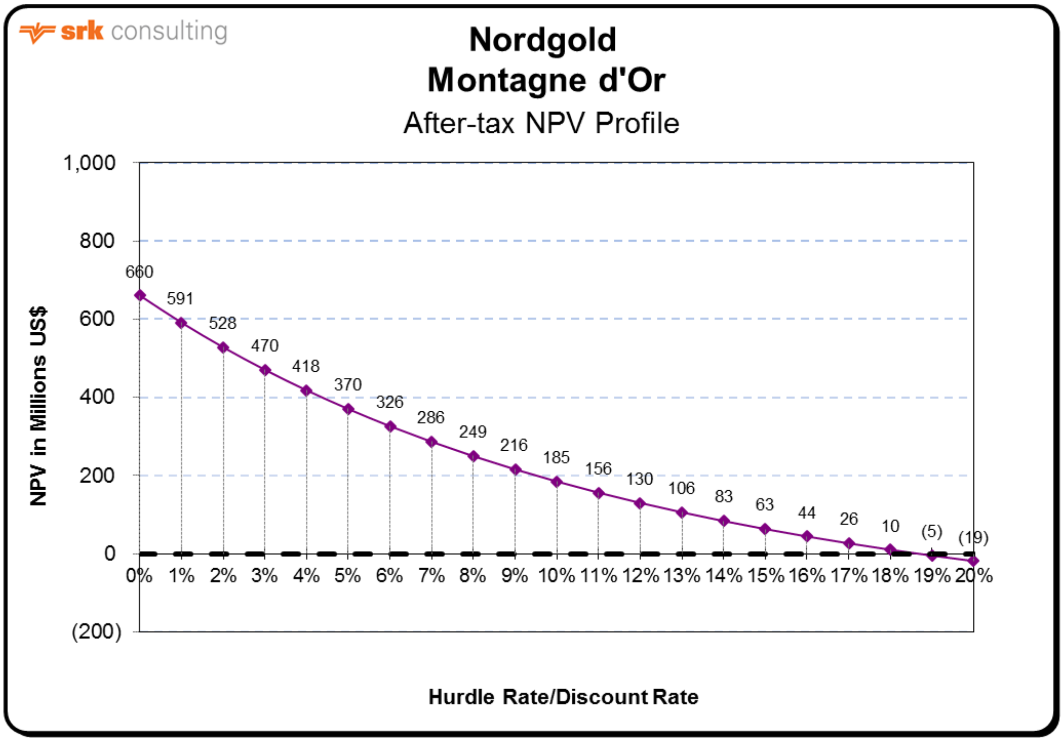
Table 22-12: Sensitivity Analysis at Various Gold Price Points

Gold Price (US$/oz)	NPV@5% (US$ millions)	IRR (%)
971	$0 (Breakeven)	5.0
1,200	307	16.8
1,250 (Base Case)	370	18.7
1,300	433	20.4
1,400	557	23.7
1,500	681	26.7

Source: SRK, 2017

22.5.2 Discount Rate Sensitivity

A sensitivity analysis of discount rates was performed by SRK due to the remote location of the Project in a jurisdiction that has little organized mining activity. Figure 22-3 shows that the Project as currently modelled would be NPV positive through an 18.5% discount rate.



Source: SRK, 2017

Figure 22-3: Project NPV Sensitivities at Varying Discount Rates

22.5.3 French Overseas Department Tax Credit Program

As currently designed and in the current metal price and cost environment, the Project requires significant Overseas Department tax credits to achieve a reasonable return on investment. This situation is highlighted in Table 22-13 which shows the results of the base case which assumes the program would continue through the LoM of the Project past its current 2020 expiry date compared to various levels of tax credit participation. At the extreme, there is a 45% decrease in Project IRR from the base case with full utilization compared to a scenario when they are not used. Given the size of the Project, it is certain that the tax credit will be subject to a prior approval to be given in advance by the French central Tax Authorities.

Table 22-13: Sensitivity Analysis at Various Tax Credit Levels (US$ millions)

Tax Credit Level	Tax Credits Generated	NPV 5%	IRR (%)	% Var from Base Case IRR
LoM (Base Case)	238	370	18.7	-
5 Yr Extension which ends 2025	207	350	18.2	-2.7
Ends 2020	115	272	14.9	-20.3
Not used	-	166	10.3	-45.0

Source: SRK, 2017

23 Adjacent Properties

There are no adjacent properties.

24 Other Relevant Data and Information

24.1 Project Implementation

An implementation plan was developed utilising a combination of owner 'self-perform' and EPCM managed contractors. This plan, which will be finalized and detailed by the EPCM Engineer during the front end engineering and design phase of Project implementation, will form the basis of the execution philosophy going forward and will be reviewed at regular intervals and revised as appropriate during Project implementation.

The wet season will potentially have its greatest impact on the earthworks based scopes, i.e. access road and site bulk earthworks including TSF construction. With this in mind the Project schedule has construction of the Apatou – Citron access road upgrade commencing the first dry season of the schedule, June/July 2019.

A preliminary Project schedule has been developed based on commitment of funds for engineering, procurement and site works, all permits will be in place and work commencing January 2019 (Table 24-1).

Table 24-1: Key Project Milestone Dates

Event	Date
Project permits and approvals in place	December 2018
Award EPCM contract and commence engineering and commitment of funds	January 2019
Pioneer 'fly camp' ready for occupation	June 2019
Access road rehabilitation commences	July 2019
Site clearing commences	July 2019
Site earthworks commences	September 2019
Permanent power supply construction commences	November 2019
Mobilize pioneer mining fleet	January 2020
Plant site earthworks commences	April 2020
Main camp ready for full occupation	May 2020
Power on HV substation	May 2021
Process plant practical completion	Oct 2021
First gold	4Q 2021

Source: Lycopodium, 2016

The schedule is highly dependent on the rapid development of site access and site infrastructure to support early construction activities. These early works include:

- Mine pre-production and accessing waste rock for TSF and plant site construction;
- Upgrading of the site access road including resurfacing the road, new bridges at river crossings, and general drainage and water management infrastructure;
- Upgrading of the existing Camp Citron to provide pioneer accommodation on site for early site works;
- Early earthworks for site roads, accommodation camp, laydown areas and sediment control facilities to protect local creeks and rivers;
- Clearing of the rainforest that covers much of the Project site;
- Progressive erection of the permanent camp to provide accommodation capacity as site works ramp up;
- Establishment of a temporary fuel farm and power station;

- Establishment of a logistics system capable of providing fuel, food, materials and manpower movement to support the construction effort; and
- Preparation of site and services for construction offices for owners, engineers and contractors.

25 Interpretation and Conclusions

25.1 Geology and Resources

Geology and resources interpretations and conclusions are:

- Columbus has completed an industry standard exploration drilling program over an area of approximately 1 ¼ km^2;
- The average drill spacing is approximately 35 m x 50 m in the measured resource, 50 m x 75 m in the indicated resource and 100 m to 150 m in the inferred resource,;
- The exploration work has been accompanied by an industry standard QA/QC program showing high quality test results;
- Columbus has conducted extensive core logging resulting in a high quality geologic model,
- The results of the drilling, sampling, analytical testing, core logging and geologic interpretation provide good support for an industry standard resource estimation; and
- The results of the Mineral Resource estimation confined within a Whittle™ pit shell optimization, hosts a Measured and Indicated Mineral Resource of 85 Mt at an average Au grade of 1.4 g/t containing 3.9 Moz of gold and an additional Inferred Mineral Resource of 20 Mt at an average Au grade of 1.5 g/t containing 1.0 Moz of gold.

25.2 Geotechnical

The completed field investigation program was conducted using accepted industry standards and procedures. The data collected is sufficient for a BFS level design. Stability of the overall pit slopes has been demonstrated using industry accepted slope acceptance criteria.

A slope monitoring program should be implemented before mining and earthworks on the Project site. The slope monitoring program will be used to identify any incipient failures and determine the course of action, which could include unloading or buttressing of slopes if a slide or failure is identified.

Two major geotechnical domains were identified in the Project. A hard rock slope composed of strong foliated metamorphic rock and a near surface saprolite soil domain that controls the stability of the upper 30 to 40 m of the ground. The saprolite is a deeply and intensely weathered residual rock that behaves like a soil. It is weak, nearly saturated, and easily deformable.

When the saprolite cuts are exposed at the recommended interramp angle of 30° they will be subjected to deformation, erosion, and failure mechanisms. Even though the saprolite slope cuts have been designed to meet the slope acceptance criteria at a FoS of 1.3, some slope failure mechanisms might occur that are not addressed by stability analysis. These failure mechanisms include gullying, piping, and erosion. These mechanisms will be exacerbated by precipitation onto exposed slopes that have not been sufficiently revegetated. Therefore, vegetative cover should be established on all cut slopes following excavation prior to the main rainy season. Berm surfaces should be graded at 2° to 3° to assist drainage off benches.

The fresh hard rock consists of granodiorite, felsic tuff, mafic volcanics, and diabase dikes. Structural features (discontinuities) encountered during this field investigation consisted of joints, lithological contacts, veins, dikes, foliation, faults, shear zones, and fractures. The rock is characterized as strong to very strong with UCS values ranging from 80 to 200 MPa. The rock is moderately jointed and has a very strong foliation joint set dipping south at approximately 70° throughout the deposit. RQD values are in the 90's and the rock mass rating ranges from 50 to 70, which indicates a fair to good quality rock mass.

A structural model was developed for the Project (Benn, 2016). A total of 23 major structures were modeled using LeapFrog® software (ARANZ Geo Limited, 2014). These include two primary fault orientations that are near vertical and two sets of shear zones. The shear zones are geologic shear zones and consist of fresh strong rock as described above.

25.3 Mineral Reserve Estimate

The mine block model, geotechnical stability, pit design, phase design, dump design, production schedule and reserve estimation have been completed to BFS level. The Project as defined in the in BFS generates a positive cash flow using only Measured and Indicated resources for the conversion of reserves using a US$1,200/oz gold price. The mine design supports the style and size of equipment selected for operations with weather corrections applied to various months of the year accounting for the tropical and potentially wet periods of time. While subject to continual improvement, the mine plan implementation will require qualified staff and the integration of all mining and related disciplines for the successful execution of the Project.

25.4 Mining Methods

The following items are SRK's interpretation of the potential opportunities and risks associated with the Project:

- If there is expansion of mineralization to the east and west of the deposit, there is the possibility for lower strip ratio available during the payback period of the operation. This may also work in conjunction with a continued higher saprolite component of the mill feed, potentially increasing plant throughput;
- The orebody, swells, contracts, splits and has geological features controlling mineralization. As a result, it is likely that grade control will be relatively difficult and variable. By the nature of block modelling and the SMU size used in the mine production schedule, a smoothed estimate to best represent the exploration drilling has been conducted through resource grade estimation. If during operations, the grade control system can be optimized from the default blast pattern incorporated into the BFS, it may be possible to further separate high grade portions of the deposit and reduce dilution. Possible optimizations may include angled production probe drilling and modelling trench sampling perpendicular to the grade orientation;
- As with grade control and associated ore mining operations being a potential opportunity to extract additional value from mining, dilution may also be a risk factor if the deposit proves to be more variable than captured in the mine block model, or if ore mining operations are not run efficiently. While no true reconciliation is practically possible until operations commence, if dilution is considered to be a potential risk then a reduced pattern drill program may be instituted to further estimate, prove or disprove local grade variability on a small scale of the deposit before mining commences;

- The pit design uses drilling that was cited for resource expansion rather than placed to confirm/ support the dimensions of the reserve pit. A drilling campaign to confirm the location of the pit toe on the high wall side of the pit will enable engineers to optimize the final pit design potentially straightening pit walls, removing low points in the pit floor, and optimizing the final strip ratio on the most sensitive slope with greater accuracy. This will provide the potential to strongly control the final strip ratio for the Project (and economics of the pit), improve geotechnical related design, and potentially reduce overall complexity in constructing final pit walls;
- The mine production rate is reasonably high and this combined with mining multiple faces, nature of the ore and pioneering required to access hard rock, operator training, mine planning and grade control will be high priorities in delivering targeted ore tonnage and grade each year. Given the relative lack of previously skilled mining personnel in French Guiana, the mine operators will need to be diligent in hiring and training personnel to operate and overcome inherent challenges in the operations; and
- The water management of the saprolite benches will be important to reduce water inflow into the pit, but also to reduce the possible instability of the saprolite in saturated conditions. Of a particular concern will be the saprolite/hard rock interface where 30 m pit faces will convert to 5 m saprolite benches. While this will be addressed with risk reduction features such as radar surveillance, the maintenance and constructability of the saprolite pit cuts will fall to the mining operations.

25.5 Mineral Processing and Metallurgical Testing

Metallurgical and processing interpretations include the following:

- The BFS metallurgical program focused on the development of a process flowsheet that included gravity concentration followed by cyanidation of the gravity tailings and intensive cyanide leaching of the gravity concentrate;
- The metallurgical program was conducted on three master composites, 15 variability composites representing different ore lithologies and grade ranges, and seven variability composites representing seven mining phases that were identified at the start of the program;
- Montagne d'Or ore can be readily processed to recover the contained gold and silver values using unit operations considered standard to the industry;
- SRK has estimated overall adjusted gold and silver recoveries based on the contribution from each ore lithology during each phase of mining. During the first six mining phases gold recovery is estimated at 94% to 95% and silver recovery is estimated at about 54% to 56%. These recovery projections include a 2% deduction from reported laboratory test results to account for inherent plant inefficiencies; and
- Detoxification of the cyanide leach residues was accomplished with the industry-standard SO_2/Air process. It was demonstrated that cyanide in the leach residue could readily be detoxified to less than 1 ppm CN_{wad}. SO_2 consumption in the range of about 5 to 6 g SO_2/g CN_{wad} were reported, which is typical of industry practice.

25.6 Recovery Methods

The selected flowsheet incorporating single stage SAG milling, gravity separation, a hybrid CIP/CIL circuit, elution and cyanide detoxification is appropriate for the demonstrated properties of the Montagne d'Or ore types, conventional and is considered low risk.

The level of test work and engineering development performed are appropriate for a BFS and support the capital and operating cost estimates developed.

25.7 Project Infrastructure

While existing infrastructure at site is suitable to support a limited exploration and drilling program, it is inadequate to support construction or ongoing operation of a project as defined in the BFS. However, none of the current logistical or regional capacity/capability limitations are incapable of being overcome and their limitations have been factored into the development of the study.

The Project schedule is dependent on the provision of all-weather site road access and establishing sufficient site services to support early construction activities (forest clearing, earthworks and early mining activities). This requires particular attention during the early planning phases and would benefit from the early commitment of funds to upgrade the road access.

It is understood from the regional power authority that the provision of grid power is dependent on a yet to be constructed power station panned for the region. If the Project is to be reliant on grid power, then a firm commitment and compliant schedule for construction must be obtained.

25.8 Tailings Storage Facility

SRK developed a phased TSF design which contains approximately 56 Mt of tailings, corresponding to approximately 12 years at a rate of 4.6 Mt/y, and follows the French Guiana requirements for BAT, with key technical issues presented as follows:

- The design considers construction, operations and closure;
- The TSF embankment will be expanded in stages using the downstream construction method;
- A geomembrane liner will be installed between the tailings and foundation soils to minimize seepage;
- Slurried tailings will be discharged from the perimeter of the TSF embankments, creating a low point/supernatant pool against native ground;
- A site specific water balance and hydrology (surface water and spillway) design has been developed;
- The TSF has been designed with a minimum stability FoS of 1.3;
- Supernatant (process water collected in the TSF) will be collected and used for process make up water as much as possible;
- A diversion channel will be constructed around the perimeter of the TSF, sized to convey the 10-year, 24-hour storm event. Storm events in excess of this event will be diverted into the TSF; and
- The closure spillway has been sized to convey the PMP event.

25.9 Site Water Management

Stormwater Management

The Project is located in an area of high rainfall, therefore it is anticipated that the system will consistently experience high intensity short duration stormwater. Additionally, low intensity contact water inflows will result in a steady inflow of water to the mine facilities.

Site water management at the Project includes management of stormwater run-off at the site and the management of the accumulation and consumption of contact and process water within the mine facilities.

Stormwater is addressed by diverting run-on to the Project around the facilities so that it remains non-contact water. The non-contact diversion system includes almost 15 km of ditches, road side channels, and diversions around the WRDs, pits, stockpiles, and TSF. Some of these diversions will be covered as the Project facilities expand and will be reconstructed as needed in response to the facility growth. In addition, seven sediment control ponds have been located around the Project, downstream of the diversions to collect and control sediment laden waters released from the site to prevent non-compliant sediment releases.

Water management of run-off and seepage from the active WRDs will be addressed through internal channels and collection ponds that will route water to a low point adjacent to the dump for collection.

When contact water is generated at the site, it is routed to the CWP where it can be stored for future use as Project makeup. However, the water balance modeling indicates the system will consistently run positive and excess contact water must be discharged from the system in order to prevent uncontrolled releases. Water balance modeling indicated that treatment and discharge from the system at a rate of up to 180 L/sec is required to maintain a net neutral or net negative balance in the system. The volume of water treated on an annual basis ranges from approximately 0.82 Mm^3 during the first year of mining to 5.68 Mm^3 during the last year of mining. Treatment needs decrease during the last years of the Project as mining ceases while milling continues and the pit is allowed to flood, decreasing the inflow of contact water to the CWP.

The process water circuit will also generate net excess water as a result of precipitation inflows to the TSF. Water balance modeling indicated that treatment and discharge from the system at a rate of up to 140 L/sec is needed to maintain a net a net neutral or net negative balance in the system from TSF Phase 2 onwards. Water treatment is not needed during the first two years of TSF Phase 1, but after that, the annual volume of water treated ranges from approximately 0.85 Mm^3 during the third year of mining to 2.88 Mm^3 during the closure activities. The water balance model predicted that water within the TSF will continue to accumulate after processing has ceased until a closure cover can be installed on top of the TSF. The water balance model assumed that once the closure cover was completed on the TSF, clean, non-contact water could be discharged from the TSF to the environment.

Because the long and intense wet season of the region, surface water inflows to the pit, both from run-off from the exposed pit walls and run-on from upgradient areas that cannot be feasibly diverted around the pit, will report to the pit bottoms along with groundwater inflows and will accumulate until it can be evacuated by pumping to the pit rim and then to the contact water management system. The mine water management plan includes a pumping system designed to evacuate the pit bottoms as rapidly as possible, but accumulation in pit sumps during the wet season is unavoidable. Mining activities should incorporate contingency plans to address the possibility of pit flooding and heavy run-off on the pit walls during periods of intense rainfall.

When mining ceases, the open pit will fill with a combination of groundwater and a predominant amount of run-off and direct precipitation. The initial groundwater contribution will be about 40% of the total inflow. Groundwater inflow will decrease as the lake fills, and will comprise a small component of inflow once the pit lake reaches the overflow point. Once the pit lake reaches the overflow point of the pit, it will be routed to undisturbed drainages, as the pit lake water quality is expected to be suitable for discharge.

25.10 Environmental Studies and Permitting

A number of technical environmental studies have been conducted as part of Project development, many of which were prepared as part of the Montagne d'Or Gold Project Environmental Scoping Study (WSP, 2015). These studies are intended to provide direction for the environmental assessment process, and guide the environmental authorities with the information required to determine the range of information and degree of detail needed in the formal impact assessment. WSP (2015) provides a preliminary identification of the regulatory elements to which the Project is subject, based on information currently available.

In addition to the land restrictions presented by the SDOM, the Project is located adjacent to a nature reserve, the Integral Biological Réserve Lucifer Dékou-Dékou, managed by the French National Forestry Board (ONF). The Montagne d'Or deposit itself is within a zone where open pit mining is permitted and the outer limit of the pit design is located at least 440 m from the reserve boundary. Consideration will be given to the proximity of the mine during the permitting process.

French Guiana's mining regime is governed by the legislative and regulatory regime applicable to the French mainland with the exception of certain legal and regulatory provisions which are specific to it in order to take into account particular characteristics and constraints of this overseas territory. The French Environment Code has specific regulations for facilities (including mining operations).

Typically, the permitting process for mines of this size and nature is initiated at the PFS state; however, the permitting process is dependent upon this BFS to start that process. It is currently envisioned that the Montagne d'Or permitting process will require at least two years to complete for the mine, plant, and explosives emulsion plant. Each major permit application must include an EA which includes Avoid-Reduce-Compensate measures, and a specific focus on endangered species; a HS evaluating major risk scenarios for the Project define preventive and protective measures; as well as relevant technical studies supporting the findings of the EA and HS.

The objective of reclamation activities will be to provide long-term stability, waste containment (to avoid both migration of pollutants and waste and minimize the risk of oxidation, leachate generation, and release of heavy metals), and erosion prevention to reduce impact on the environment. Following the development of the environmental and social impact assessment, and associated environmental management plans, Nordgold may have an opportunity to modify the closure approaches during detailed design when more information has been developed, and equivalent levels of environmental protection can be effectively demonstrated.

25.11 Capital and Operating Costs

Total capital costs totaling US$827 million including final closure/reclamation costs are summarized in Table 25-1. Approximately 9.5% overall contingency has been applied to capital items, which is appropriate for a BFS level of analysis in SRK opinion. The initial capital required to construct a 4.6 Mt/y project that will produce approximately 237 koz/y during the first 10 years of the operation is estimated to be US$535.2 million which includes US$52 million of preproduction costs, all of which are based on an EURUSD of US$1.05:€1.00.

Table 25-1: Life-of-Mine Capital Costs (US$000's)

Description	**US$ @ 1.05**
Initial Capital Costs	
Preproduction Costs	52,003
Mining	69,047
TSF/Process/Infrastructure	403,991
Water Management	10,150
Total Initial Capital	**$535,191**
Sustaining Capital Costs	
Mining	61,208
Process	-
Infrastructure	13,477
TSF	151,282
Water Management	5,154
Total Sustaining Capital	**$231,120**
Total Capital Costs	
Preproduction Costs	52,003
Mining	130,255
TSF/Process/Infrastructure	403,991
Infrastructure (Sustaining)	13,477
TSF (Sustaining)	151,282
Water Management	15,304
Subtotal Capital Costs	**$766,312**
Closure/Reclamation	60,659
Total LoM Capital Costs	**$826,971**

Source: SRK, 2017

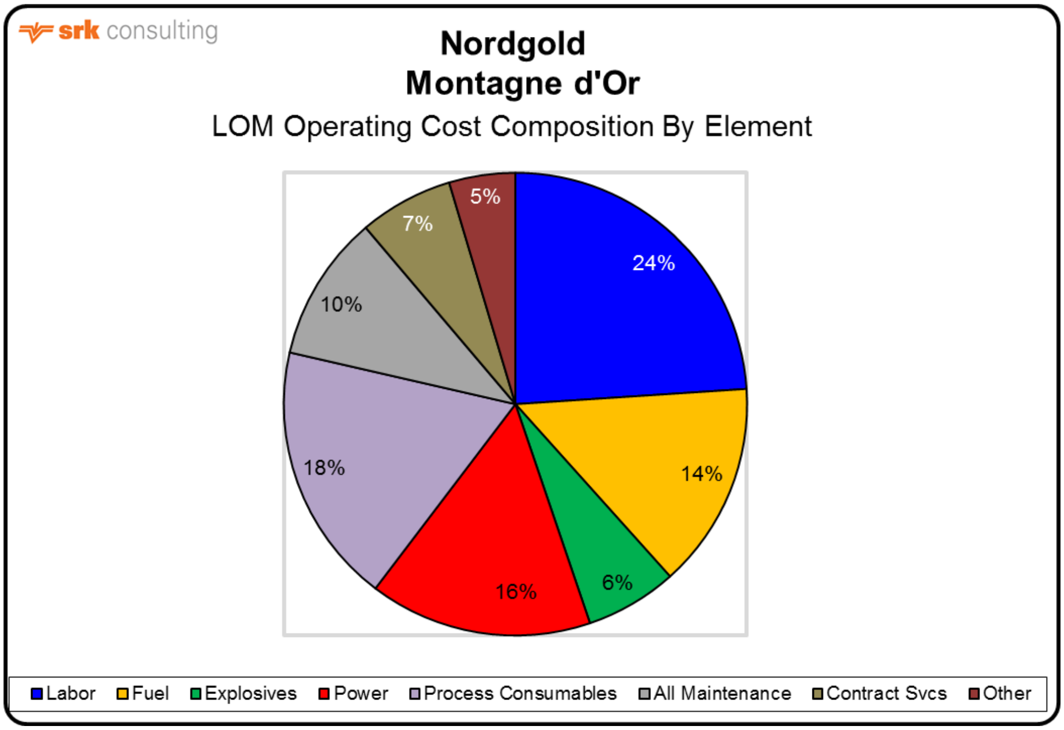
Table 25-2 presents LoM operating costs of US$28.76/t processed used in the TEM, all of which are based on an EURUSD of US$1.05:€1.00.

Figure 25-1 shows the relative composition of operating cost inputs such as labour, process consumables and power which are the three largest cost items at 24%, 18%, and 16%, respectively.

Table 25-2: Operating Cost Summary (US$000's and US$/t Processed)

Description	**US$ @ 1.05**
Mining	704,040
Process	621,830
Site G&A	224,309
Water Management	6,368
Total Operating Costs	**$1,556,547**
Operating Cost Unit Rates	**US$/t Proc.**
Mining ($/t mined)	2.44
Mining ($/t processed)	13.01
Process	11.49
Site G&A	4.15
Water Management	0.12
Total Operating Costs	**$28.76**

Source: SRK, 2017



Source: SRK, 2017

Figure 25-1: Operating Cost Composition

25.12 Economic Analysis

The indicative economic results summarized in this section are based upon work performed by SRK, Lycopodium or received from Nordgold in 2016. They have been prepared on both an annual pre-tax and after-tax basis, a 100% equity basis with no Project financing inputs, and utilizes a 1.05 EURUSD.

The project design is a 4.6 Mt/y operation that would cost an estimated US$535 million to build. The project is expected to produce 214 koz Au per year at an AISC of US$779/oz (including the first 10 years producing 237 koz Au per year at an AISC of US$749/oz). Project metrics are summarized in Table 25-3 and show a NPV 5% value of US$370 million and 18.7% IRR. This valuation is helped in large part by French government surplus tax credit refunds of US$186 million during 2020-2023. At the extreme, there is 45% decrease in project IRR from the base case with full tax credit utilization compared to a scenario when they are not used. In addition to the impact of the tax credits, the project is most sensitive to gold grade (and price), EURUSD, operating costs and capital costs.

Table 25-3: Project Valuation Summary (US$000's)

Description	US$ @ 1.05
Net Revenues	$3,058,905
Operating Costs	(1,556,547)
Operating Margin	$1,502,358
Income Taxes	(200,746)
Operating Cash Flow	$1,301,612
Initial Capital	(535,191)
Sustaining Capital	(231,120)
Closure/Reclamation Capital	(60,659)
Total Capital	($826,971)
Surplus Tax Credit Refunds	185,632
Free Cash Flow	$660,273
NPV 5%	$369,949
IRR	18.7%
AISC	US$779/oz

Source: SRK, 2017

26 Recommendations

26.1 Geology and Resources

SRK has the following recommendation regarding geology and resources:

- Additional infill drilling at the 35 m x 50 m spacing could be completed in the areas of early mining to provide additional confidence in the tonnes and grade of this production;
- Infill drilling is recommended to target the areas where Inferred Resources are located within the Reserve pit where the current resource Au block grades are estimated to be above mining CoG. This could in turn convert current Inferred Mineral Resource to Mineral Reserves; and
- Additional sample analysis could also be conducted to refine the current NAG and PAG model.

26.2 Geotechnical

SRK recommends the following systems for costing as a part of the BFS:

- Slope Radar (two units are recommended) for monitoring the active mining areas on the north and south walls of the pit. These monitoring units should be initially targeted on saprolite slope cuts. One radar unit may suffice for the first 2 to 3 years of mining. Radar monitoring is recommended at the onset of mining due to the existing mapped landslides (Rostan, 2015) on saprolite slopes above the open pit;
- Radar monitoring is recommended at the onset of mining;
- Robotic total stations and prisms should be placed for long-term slope monitoring. An initial layout of prisms every 200 m along the crest of the slope and can be used for long-term monitoring. This is recommended at the top of the saprolite and the top of the hard rock contact;
- Two to three wire extensometers should be kept at the site and utilized when tension cracks or the initial signs of a failure are observed;
- InSar monitoring is recommended for the Project to monitor long-term movement, on a monthly basis, of the saprolite slope above the pit and towards the national park boundary; and
- A geotechnical engineer should be engaged on the Project for both slope monitoring and to assist the mine engineering staff with the operation of the mine.

SRK recommends Nordgold implement a slope monitoring program prior to the beginning of mining and earthworks on the Project site. The slope monitoring program will be used to identify any incipient failures and determine the course of action, which could include unloading or buttressing of slopes if a slide or failure is identified.

Utilize InSar monitoring data for the native slopes south of the pit during mining. There is a precedence for large failure events occurring on existing saprolite slopes within the Project without mining (Rostan, 2015). The natural slope saprolite hazard will remain, but any incipient failure will be identified by InSar monitoring. This monitoring will be used to decide on a course of action if an incipient failure is identified.

WRDs should include a coarse underdrain material, which is a minimum of 5 m thick, following the course of any existing drainages. The coarse underdrain may be constructed of RoM waste.

Bench face and slope performance should be assessed to determine if there is opportunity to optimize the slope angles. The pit slope design meets industry slope acceptance criteria at a feasibility-level. Principal validation of slope angles with be through slope performance and rock fabric mapping of the exposed pit walls. Mapping of the hard and fresh rock slopes should be completed as the pit progresses to collect joint set length and additional spacing data.

Ensure adequate drainage design and vegetative cover on saprolite slopes is designed and implemented in the final Project plans. Effective drainage of the saprolite soils is critical to the Project. When the saprolite cuts are exposed at the recommended interramp angle of 30° they may be subjected to the displacement, erosional, and failure mechanisms if adequate drainage and vegetative cover is not designed and constructed. The saprolite slopes, as designed, meet the slope acceptance criteria at a FoS of 1.3, however potential failure mechanisms may occur including gullying, piping, and erosion. These failure mechanisms will be exacerbated by direct precipitation onto uncovered slopes. Therefore, vegetative cover should be established on all cut slopes following excavation prior to the rainy season. Slopes should be graded at 2° to assist drainage off of the slopes.

26.3 Mining Methods

SRK recommends the following during the detailed engineering, or as part of mining engineering work during the pre-production mining period:

- A fence of drillholes target the pit toe on the south wall;
- An additional condemnation drilling program to confirm some of the infrastructure locations should be assessed, and suitable holes planned as required;
- Additional data collection, site investigations for foundations, and related studies, are recommended to advance the level of design with respect to the geotechnical aspects for the WRD and stockpile locations, to demonstrate that a soil/geomembrane barrier system is not required at the WRD; and
- Additional mining related studies be performed for the detailed design that include the following:
    - Detailed scheduling for pre-production earthworks;
    - Continued discussion with vendors for equipment quotes;
    - For detailed engineering, the low-grade saprolite stockpile design should be advanced;
    - Enhance understanding of WRD foundation conditions concerning suitability of compacted saprolite as a control for WRD seepage;
    - Development of operational guidelines for treatment of ARDML waste rock. Customization of rapid PAG field testing would also be advised; and
    - An infill drilling program to optimize mine design related to the pit toe of the reserve pit, internal waste intrusions, saprolite/hardrock interface and grade variability.

26.4 Mineral Processing and Metallurgical Testing

The metallurgical programs completed are considered adequate to move the Project forward and there are no recommendations for additional studies to further evaluate the currently defined processing methodology.

26.5 Recovery Methods

The process plant design is based on appropriate design criteria and a robust flowsheet with minimal requirements for further development prior to Project implementation.

Allowance has been made in the earthworks estimate to undertake additional investigations at the proposed plant site to confirm geotechnical conditions prior to completing the corresponding foundation design. The existing design is based on pile foundations beneath critical structures, which may not be necessary subject to the confirmation of subsurface conditions. There may, therefore, be an opportunity to reduce the capital cost for the process plant foundations and it is recommended that these investigations commence once Project permitting is complete.

Additional field investigations are required prior to final plant foundation design. Drilling complemented by Standard Penetration Tests (SPTs) and cone penetration tests (CPTs) is recommended to confirm foundation conditions for final design. The additional field work should consist of 20 to 30 holes with SPT logging and, where appropriate, CPT probes located within the foundation footprint. The number of drillholes may be reduced if geophysical surveys of the saprolite, saprock and bedrock contacts can be successfully completed. Additionally, a geophysical investigation should be conducted to determine dynamic soil properties. Additional geotechnical characterization, laboratory and field testing of the saprolite soils, and the potential need for planned additives such as waste rock and/or lime should to be conducted to provide data to in bring cost estimates to a final design level.

26.6 Project Infrastructure

Due to the remoteness of the site there is little reliance on local infrastructure other than the logistical requirements to move material through the ports and on regional roads. Most other infrastructure including water supply, accommodation, offices and workshops and waste disposal facilities will be provided by the Project.

The tie in of the Project to the national power grid is dependent on the grid having the capacity to support the additional load. While the power authority had indicated that this will be the case with the development of a new regional power station to boost supply, the viability of the Project as currently envisioned is currently dependent on the development of that power station.

The fall-back position is to install a power plant at site. While the capital cost of this alternative is actually lower than the construction of the overhead powerline the cost of power generation is significantly higher than the grid tariff quoted.

It is recommended that a firm commitment be obtained from the power authorities to meet the needs of the Project before work commences on construction.

26.7 Tailings Storage Facility

SRK recommends that the following tasks be considered to reduce the uncertainty associated with the BFS TSF engineering design:

- The current design was based on SRK's and Nordgold's understanding of French permitting requirements being applied to French Guiana. The final design should address permitting requirements once permitting approvals have been issued;
- All water input and output values should be recorded throughout the operational life of the TSF and the water balance model updated on an annual basis;
- Additional field investigations should be performed in the TSF footprint areas, including supplementary characterization of the foundation conditions, tailings material, and potential borrow areas (i.e. at a detailed engineering level);
- Site specific daily climatological data (precipitation, evaporation, minimum/maximum temperatures, etc.) should be collected and compared against the synthesized data used in the BFS design;
- The anticipated tailings supernatant geochemistry should be reviewed and the assumption that supernatant can be recirculated through the process and the tailings area net AGP should be confirmed;
- A risk assessment for the TSF should be completed to confirm the final design scope and design parameters;
- In lieu of a spillway for TSF Phases 1, 2 and 3, storage for the PMP event was included. Future designs should evaluate operational spillways for each phase versus design storm volumes anticipated in the TSF and the associated freeboard requirements;
- A monitoring program, including piezometers, survey monuments and groundwater monitoring wells should be established as part of detailed design. The program should also include annual reviews, independent audits and Safety evaluation of existing dams (SEEDs) to be developed as part of the final design;
- The closure design should be reviewed, and if necessary, updated during the detailed design, taking into consideration regulatory requirements; and
- An operations, maintenance and surveillance (OMS) manual, which guides the operation of the TSF, should be developed as part of detailed design, and include such items as:
    - The supernatant pool should be kept as far from the embankments as possible at all times;
    - A detailed Project construction schedule should be developed that considers the contractor equipment, earthwork quantities (including wastage) and dry/wet seasons;
    - The use of an observational approach to provide an understanding of the actual performance of the facility should be implemented during operations. The periodic review of the performance of the facility should be accomplished in light of field observations to provide guidance for future operations. Operations personnel should closely monitor the observed seepage, pore pressures, and phreatic surface. Refinements and modifications to the design and operational procedures should be made based on observed conditions and monitoring data, as appropriate; and
    - Because the TSF is assumed to be lined with geomembrane, the stability analysis assumed that there will be no phreatic surface within the embankment. However, an

elevated phreatic surface, from a leak in the liner system or an elevated ground water surface, could adversely affect the TSF embankment stability. SRK recommends that piezometers be installed in the embankment to monitor potential phreatic surfaces within the TSF embankment to confirm that elevated phreatic conditions are not being developed, and underdrains be evaluated in the TSF design, to reduce the potential to develop elevated phreatic surfaces.

26.8 Site Water Management

26.8.1 Hydrogeology

From a hydrogeology standpoint, dewatering of the open pit is driven by surface water run-off, rather than by groundwater inflow. As a result, it is unlikely that active dewatering of the bedrock or saprock with wells around the pit perimeter will reduce costs or significantly improve long term mining conditions in the open pit. Therefore, no additional dewatering-related work is recommended. However, the following recommendations are appropriate for assessing long term impacts and for monitoring water levels as mining begins:

- Continue the creek flow accretion monitoring on Apollon and Infirmes creeks. Analyze the data acquired between August of 2016 through August of 2017 on a continuous basis and make interpretations on baseflow in the creeks;
- After a set of flow accretion data from a full dry season has been analyzed, recalibrate the numerical model to observed baseflows. Reassess impacts to the high creeks in the RBI, and specifically Apollon Creek; and
- Prior to mining, add a set of three nested piezometers or observation wells above the pit perimeter, and along the full extent of the pit rim. Complete either nested vibrating wire piezometers or nested standpipe wells in the saprock and bedrock, respectively, at 3 locations. These installations will allow Nordgold to assess the materials above the pit for geotechnical stability. Furthermore, they will allow Nordgold to track dewatering progression above the pit, between the mining operation and the RBI.

26.8.2 Surface Water

From a hydrology standpoint, the site has a great capacity to produce high volumes of run-off that can have significant impact on mining activities. Mine water management involves both diverting storm flows around the facility to prevent damage from erosion and to avoid uncontrolled discharges of water that has come into contact with mining activities. The following recommendations are provided to increase the understanding of the hydrology regime and improve the management of water at the mine site:

- Management of the TSF supernatant pool is limited to a narrow range during operations, with the intent of maximizing the area of exposed beach to enhance consolidation, and to provide a large surge capacity to contain the inflow from extreme storm events. Maintaining such tight control will require diligent monitoring of the TSF pool and establishing of reliable method of predicting inflows. The system should be prepared to address the possibility of high rainfall at any time during operations that will result in unexpected inflows to the TSF water management system;

- Similarly, the Contact Water Management system must maintain a delicate balance between ensuring sufficient water is available to sustain operations during an extreme drought, while at the same time maintaining sufficient surge capacity within the CWP to contain the inflow from extreme storm events. Criteria by which the pool is managed, begun in this study, must be expanded as the understanding of the Project expands;
- Design elements for the Sedimentation Ponds and the CWP will need to be included. Design elements to include intake and outlet control structures, erosion management, excavation and grading. Designs are required prior to finalizing the position of the water management diversion ditches and energy dissipation structures. Detailed engineering of the mine water management components will be required to advance this Project to design level;
- SRK is aware of continued climate and streamflow monitoring at the site. This data should be used to regularly update the understanding of the climatic conditions and hydrological behavior at the site. Refinement of these behaviors could have significant ramifications on mine water management at the site; and
- The tropical environment at the Project will necessitate regular maintenance of all diversion ditches and sediment ponds. Heavy sediment loads and rapid vegetation growth can be expected, which will require a regular schedule of cleanout and clearing of the mine water management infrastructure. Many of these structures will be remote from the regularly travelled portions of the mine site and frequent inspection of the diversion channels and sediment control ponds is critical. Additionally, an evaluation of the maintenance needs of the Project should be performed regularly to ensure the sufficient resources are available.

26.8.3 Geochemistry

Contact water from the felsic tuff and lapilli tuff will eventually become acidic with elevated metals and will likely require management. A closure strategy of cover emplacement concurrent with waste rock deposition, in conjunction with a material handling and segregation plan, could significantly attenuate the production of acid rock drainage from waste rock.

At closure, the recommended reclamation option is to cover and revegetate the WRDs to stabilize the dumps, minimize infiltration of water and oxygen, and construct drainage pathways that direct run-off away from the dumps. Isolation of the identified PAG rock from water and oxygen early on during operations is highly recommended, with the objective of precluding the accumulation of acidic pore water in the dump that could persist into closure and create a scenario in which draindown of acidic, metal-laden water is released to the environment slowly over time.

SRK recommends that additional data be collected to supplement the current PAG-NAG block model. Design and execution of a material handling and segregation plan should be a priority. This must involve real time field sampling (blastholes and/or pit benches) and analysis of NAG-PAG geochemical properties with rapid turnaround time to facilitate material segregation and dump deposition. In the interim, another means of obtaining additional ABA data would be by collection of samples from drill core currently being stored at site.

Contact water with the pit wall rocks below the saprolite and saprock units could produce ARDML and require management. SRK recommends that the quality of this contact water should be monitored frequently during operations to determine what measures are needed. Minimizing run-on from areas outside of the pit will be important in reducing this component of contact water.

A series of long-term column leach tests is recommended to supplement the geochemical data obtained for the BFS program and provide improved confidence in the prediction of PAG vs non-PAG waste rock. The recommended test is the humidity cell (ASTM method D5744), which is the industry standard for obtaining mineral reaction data from mining waste rock and tailings. The recommended duration of the humidity cells is one year at a minimum, with a provision to continue the tests if it is determined that the tests could provide useable data. The data obtained from the tests would improve the level of confidence in discriminating PAG versus non-PAG waste rocks and would be useful in the operational waste rock segregation plan.

26.9 Environmental Studies and Permitting

From an environmental and permitting perspective, the accurate characterization of AGP of the various geological materials, and the proper management and disposal of those materials once excavated from the open pit are important considerations. SRK recommends that a detailed mine schedule be developed using the geological block model that is based on the ARDML potential of the rock, so that the deposition of these materials can be sequenced within the WRDs in a manner that places inert materials on the exterior of the facility, while sequestering potentially reactive materials in the interior. This will minimize the surface exposure of sulphidic materials to oxygen and precipitation, and allow for more effective management and closure of the WRDs, thus reducing the need for longer-term seepage monitoring and collection.

Recent clarifications from French permitting specialists suggested that the French regulations will require a numerical groundwater model be developed for the TSF, WRDs, LG ore stockpile and CWP, supported by a field characterization program and specialty geotechnical testing. Depending on the results of the program and interpretation by French regulators, there is a risk that modifications to the current containment design, currently consisting of a 2.0 mm HDPE geomembrane, may be required. If needed, potential alternatives include, for example, an overdrain and pump system to reduce the head on the liner, development of a modified soil layer as either a replacement to the geomembrane or a layer beneath the geomembrane.

Concurrent with the recent clarifications regarding seepage control measures required at the TSF by French regulators, similar discussions are ongoing for the CWP, WRDs and LG ore stockpile.

SRK recommends that a complete site-wide inventory of all potential closure cover materials be performed; that geochemical, geotechnical, and agronomical testing of these materials be conducted, and that infiltration modeling of potential cover design be completed. This will allow Nordgold to move away from the prescriptive, regulatory cover designs to more practical designs that can demonstrate equal or better protection of the environment post closure.

Addition baseline data collection will likely be required on Concession 102 (01/32), on which the proposed TSF is partially located but not currently owned by Nordgold.

26.10 Capital and Operating Costs

There are no specific recommendations with respect to the capital and operating costs at this stage.

26.11 Economic Analysis

There are no specific recommendations with respect to the economic analysis at this stage.

26.12 Recommended Work Programs

As provided by Nordgold, there exists budgeted spending of approximately US$2 million per year for 2017 and 2018 for management, environmental permitting and ongoing operations including:

- Project management;
- Regulatory and environmental specialists and consultants;
- In-country office costs;
- Public relations, community relations and stakeholder engagement programs; and
- Administration and other overheads.

(For the purposes of the BFS the budgeted costs of US$2 million per year for 2017 and 2018 were considered to be sunk costs, and were not included in the Project capital costs.)

Geology and Resources

At this time, the current drilling and resource estimate is sufficient for further advancement of the Project up to point of making a go-ahead decision.

Infill drilling is recommended to target the areas where Inferred Resources are located within the Reserve pit where the current resource Au block grades are estimated to be above mining cut off grade. This could in turn convert current Inferred Mineral Resource to Mineral Reserves.

Plant Site Geotechnical

SRK recommends completing a final geotechnical design for the plant site. The following studies and parameters should be completed and appropriate design values verified:

- A soil geophysical survey of the site should be completed to establish the bedrock depth and determine dynamic properties, including the dynamic shear modulus. This survey can also be used to determine the depth to bedrock across the plant foundation for dimensioning of pile foundations;
- CPT or SPT drilling and testing should be completed at the final foundation locations to verify soil conditions used in this analysis and to complete a final design. This is recommended as a soils rig including SPTs was not available for this program; and
- Additional testing should be completed for characterization including ASTM D4647 (pinhole test) and soil resistivity.

The cost estimate for these programs is US$130,000.

Mining and Reserves

Drilling recommendations previously mentioned are optional. Other work recommendations would be carried out as part of normal detailed engineering (part of EPCM), or as part of mining engineering work during the pre-production mining period. Therefore, associated costs for mining related programs would be already included in normal detailed engineering costs and pre-production mining costs. There are no additional costs required for the Project at this stage prior to a decision to go into construction.

Mineral Processing and Metallurgical Testing

Metallurgical testing performed to date is sufficient for advancement of the Project up to and including a decision to construct the Project.

Recovery

There are no recommended work programs required prior to a decision to construct the Project.

Project Infrastructure

There are no recommended work programs required for infrastructure prior to a decision to construct the Project.

Tailings Storage Facility

SRK recommends the following work be performed prior to the construction of the starter earthworks and the commencement of operations:

- Prior to the development of construction drawing and specifications, additional field investigations should be performed in the TSF footprint areas, including complementary characterization of the foundation conditions (i.e. where significant gaps exist), tailings material, and potential borrow areas, with an estimated cost of US$400,000;
- A field and laboratory program should be performed to characterize the in situ permeability and attenuation characteristics of the underlying saprolitic soils, as well as potential permeability amendment options for the TSF foundation soils. This data would be used to support a numerical groundwater model and demonstrate compliance with French regulations. If combined with the TSF foundation characterization program, it has an estimated additional cost of between US$100,000 to US$250,000;
- Prior to the development of construction, TSF final design drawing and specifications should be completed, which are part of the planned BFS engineering budget (subsequent to a decision to construct the Project); and
- An OMS manual which documents operations, monitoring and surveillance should be developed, and is part of the planned BFS engineering budget (subsequent to a decision to construct the Project).

Site Water Management

Recommended hydrogeology, hydrologic and climatological study costs would be covered by the planned Project permitting budget by Nordgold (for regulatory and environmental specialists), and there are no additional costs anticipated.

Detailed engineering of the mine water management components will be required to advance the Project to design level. However, these are included as part of the planned engineering budget subsequent to a decision to construct the Project and there are no additional recommended work program costs prior to a decision to construct the Project.

Geochemistry

SRK has previously noted that a series of long-term column leach tests would supplement the geochemical data obtained for the WRDs. This program is not critical for the next phase of the Project, and is not necessary for making a decision to proceed with construction of the Project. The recommended soil attenuation program for the WRD foundations is discussed in the Environmental section below.

Environmental

Recommendations regarding material excavation and sequenced disposal would be carried out as part of normal detailed engineering of the WRDs, or as part of mining engineering work during the pre-production mining period. Therefore, associated planned costs would be mainly included already in normal detailed engineering costs and pre-production mining costs.

The identification, sampling, and characterization of closure cover materials is dependent on the number of sources investigated, and could be deferred to the end of mining if not desired for initial permitting. Depending on the interpretation by French regulators, there is a risk that a materials investigation will be needed to confirm the quality and quantity of these materials.

Additional baseline data collection will be limited to the encroachment footprint of the TSF onto Concession 102, and could likely be covered by normal operating costs associated with the ongoing permitting efforts.

SRK also recommends that a field investigation should be performed within the proposed WRDs, CWP and LG ore stockpile footprint areas, to characterize the in situ permeability of the underlying saprolitic soils, foundation characteristics, and potential permeability amendment options for the foundation soils, with an estimated cost of between US$225,000 to US$625,000. In addition, materials collected during this field program would be subjected to attenuation testing with the objective of demonstrating the effectiveness of chemical constituent removal from seepage contacting and passing through the barrier systems. The chemical attenuation program has an estimated cost of between US$125,000 and US$250,000.

Capital and Operating Costs

Although no VAT is applicable in French Guiana (by exception to the other French overseas districts), the following French Guiana import taxes should be anticipated unless specific measures will be been granted to Project. In SRK's view these taxes do get commonly waived for mining projects in many jurisdictions so the subsequent risk is low but these taxes include:

- Customs duties: goods imported from third countries (outside the EU) are potentially submitted to customs duties, depending on their origin from a custom viewpoint. The rate will depend on the nature of the assets as determined by the customs tariff;
- External dock duties ("296ctroy de mer externe"): The import dock duties are due when goods (inventories or fixed assets) are imported in French Guiana from any other territory (Metropolitan France, other French overseas districts, EU Member States or third countries). They are assessed on the purchase price plus custom duties. The rate could range between 0% (many exemptions applicable) and 60%, depending on the tariff (which is 500 pages long). With respect to this case, it is anticipated that most of the assets should be subject to rates of 7.5%, 15% or 22.5%, plus a regional 2.5% duty;
- Internal dock duties ("296ctroy de mer interne"): the sale of products manufactured, transformed or extracted locally is submitted to internal dock duties, with the same rates. However, a producer submitted to the internal dock duties has a right to deduct external dock duties suffered for its production, especially when the good produced are exported. As a result only the value added is consequently submitted to the internal dock duties in such case;

- The depreciation basis of the imported assets should include both customs duties and external dock duty if not recoverable under the conditions explained above; and
- An import duty review program is recommended at a cost of US$20,000.

With respect to labour costs in French Guiana, SRK recommends the resolution of the issue to identify the impact of the benefit of some social security exemption according to a specific oversea regulation (LODEOM Renforcée) or the general French social security exemption (reduction FILLON). For the purposes of this study, Nordgold retained the less favourable scenario as it is not guaranteed that you could benefit from both the "LODEOM Renforcé" scheme up to 250 employees and the FILLON scheme for the remaining eligible employees or to obtain the benefit of the LODEOM Renforcé for all employees. A labour regulation review program is recommended at a cost of US$20,000.

Technical Economics

SRK recommends that the French Overseas Department tax credit program be evaluated in further detail due to the importance of the surplus tax credit refunds in the early part of the mine life. In particular, it would be useful to receive more information about the eligibility of preproduction costs, the TSF and the water management costs in the calculation of for the tax credit. Also, given the size of the Project, it is certain that the tax credit will be subject to a prior approval to be given in advance by the French central Tax Authorities. A tax credit review program is recommended at a cost of US$15,000.

LoM long range EURUSD forecast surveys should be done as the exchange rate has a strong impact on Project economic metrics. An exchange rate forecast program is recommended at a cost of US$15,000.

26.12.1 Summary of Recommended Work Program Costs

Recommended work program costs are summarized in Table 26-1.

Table 26-1: Summary of Costs for Recommended Work

Recommended Work Programs	Cost Estimate (US$)
In-fill Drilling on Inferred Resources within Reserve Pit	350,000
Plant Site Foundations Geotechnical Programs	130,000
WRDs/LG Stockpile Foundation Characterization Program	225,000 to 625,000
Soil Attenuation Investigation	125,000 to 250,000
TSF Geotechnical Characterization and Groundwater Modeling Program	500,000 to 650,000
Import Duty Review Program	20,000
Labour Regulation Review Program	20,000
Tax Credit Review Program	15,000
Exchange Rate Forecast Program	15,000
Total Programs	**$1,400,000 to $2,075,000**

Source: SRK, 2017

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28 Glossary

The Mineral Resources and Mineral Reserves have been classified according to CIM (CIM, 2014). Accordingly, the Resources have been classified as Measured, Indicated or Inferred, the Reserves have been classified as Proven, and Probable based on the Measured and Indicated Resources as defined below.

28.1 Mineral Resources

A **Mineral Resource** is a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling.

An **Inferred Mineral Resource** is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity. An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

An **Indicated Mineral Resource** is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit. Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation. An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probable Mineral Reserve.

A **Measured Mineral Resource** is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit. Geological evidence is derived from detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation. A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proven Mineral Reserve or to a Probable Mineral Reserve.

28.2 Mineral Reserves

A **Mineral Reserve** is the economically mineable part of a Measured and/or Indicated Mineral Resource. It includes diluting materials and allowances for losses, which may occur when the material is mined or extracted and is defined by studies at prefeasibility or feasibility level as appropriate that include application of Modifying Factors. Such studies demonstrate that, at the time of reporting, extraction could reasonably be justified.

The reference point at which Mineral Reserves are defined, usually the point where the ore is delivered to the processing plant, must be stated. It is important that, in all situations where the reference point is different, such as for a saleable product, a clarifying statement is included to ensure that the reader is fully informed as to what is being reported. The public disclosure of a Mineral Reserve must be demonstrated by a prefeasibility or feasibility study.

A **Probable Mineral Reserve** is the economically mineable part of an Indicated, and in some circumstances, a Measured Mineral Resource. The confidence in the Modifying Factors applying to a Probable Mineral Reserve is lower than that applying to a Proven Mineral Reserve.

A **Proven Mineral Reserve** is the economically mineable part of a Measured Mineral Resource. A Proven Mineral Reserve implies a high degree of confidence in the Modifying Factors.

28.3 Definition of Terms

The following general mining terms may be used in this report.

Table 28-1: Definition of Terms

Term	Definition
Assay	The chemical analysis of mineral samples to determine the metal content.
Capital Expenditure	All other expenditures not classified as operating costs.
Composite	Combining more than one sample result to give an average result over a larger distance.
Concentrate	A metal-rich product resulting from a mineral enrichment process such as gravity concentration or flotation, in which most of the desired mineral has been separated from the waste material in the ore.
Crushing	Initial process of reducing ore particle size to render it more amenable for further processing.
Cut-off Grade (CoG)	The grade of mineralized rock, which determines as to whether or not it is economic to recover its gold content by further concentration.
Dilution	Waste, which is unavoidably mined with ore.
Dip	Angle of inclination of a geological feature/rock from the horizontal.
Fault	The surface of a fracture along which movement has occurred.
Footwall	The underlying side of an orebody or stope.
Gangue	Non-valuable components of the ore.
Grade	The measure of concentration of gold within mineralized rock.
Hangingwall	The overlying side of an orebody or slope.
Haulage	A horizontal underground excavation which is used to transport mined ore.
Hydrocyclone	A process whereby material is graded according to size by exploiting centrifugal forces of particulate materials.
Igneous	Primary crystalline rock formed by the solidification of magma.
Kriging	An interpolation method of assigning values from samples to blocks that minimizes the estimation error.
Level	Horizontal tunnel the primary purpose is the transportation of personnel and materials.
Lithological	Geological description pertaining to different rock types.
LoM Plans	Life-of-Mine plans.
LRP	Long Range Plan.
Material Properties	Mine properties.
Milling	A general term used to describe the process in which the ore is crushed and ground and subjected to physical or chemical treatment to extract the valuable metals to a concentrate or finished product.
Mineral/Mining Lease	A lease area for which mineral rights are held.
Mining Assets	The Material Properties and Significant Exploration Properties.
Ongoing Capital	Capital estimates of a routine nature, which is necessary for sustaining operations.
Ore Reserve	See Mineral Reserve.
Pillar	Rock left behind to help support the excavations in an underground mine.

Term	Definition
RoM	Run-of-Mine.
Sedimentary	Pertaining to rocks formed by the accumulation of sediments, formed by the erosion of other rocks.
Shaft	An opening cut downwards from the surface for transporting personnel, equipment, supplies, ore and waste.
Sill	A thin, tabular, horizontal to sub-horizontal body of igneous rock formed by the injection of magma into planar zones of weakness.
Smelting	A high temperature pyrometallurgical operation conducted in a furnace, in which the valuable metal is collected to a molten matte or doré phase and separated from the gangue components that accumulate in a less dense molten slag phase.
Stope	Underground void created by mining.
Stratigraphy	The study of stratified rocks in terms of time and space.
Strike	Direction of line formed by the intersection of strata surfaces with the horizontal plane, always perpendicular to the dip direction.
Sulphide	A sulphur bearing mineral.
Tailings	Finely ground waste rock from which valuable minerals or metals have been extracted.
Thickening	The process of concentrating solid particles in suspension.
Total Expenditure	All expenditures including those of an operating and capital nature.
Variogram	A statistical representation of the characteristics (usually grade).

28.4 Abbreviations

The following abbreviations may be used in this report.

Table 28-2: Abbreviations

Abbreviation	Unit or Term
%	percent
°	degree (degrees)
°C	degrees Centigrade
µm	micron or microns
AA	atomic absorption
ALCATEL	Alcatel Alsthom Compagnie Générale d'Electricité
AARL	Anglo American Research Laboratory
ABA	Acid based accounting
ADT	Articulated Dump Trucks
AEX	exploitation authorizations
Ag	Silver
AGP	acid generating potential
Ai	Abrasion index
AISC	All in Sustaining Cost
ALS	ALS Metallurgy – North America
ANFO	ammonium nitrate fuel oil
ANP	Acid Neutralization Potential
ARDML	Acid Rock Drainage and Metal Leaching
ARM	Mining Research Authorizations
ASTM	American Society for Testing and Materials
Au	gold
~~AuEq~~	~~gold equivalent grade~~
~~AWBM~~	~~Australian Water Balance Model~~
Ba	barium
BAT	Best Available Technique
BFS	Bankable Feasibility study
BGM	Bureau Minier Guyanais
BOQ	Basis of Quantity

Abbreviation	Unit or Term
BRGM	Bureau de Recherches Géologiques et Minières
BV	Bureau Veritas Commodities Ltd – Inspectorate Metallurgical Division
Bwi	Bond ball mill work index
CCE	Capital cost estsimate
CGSZ	Central Guiana Shear Zone
Cr	Chromium
CIL	carbon-in-leach
CIT	Corporate income tax
cm	centimetre
cm2	square centimetre
cm3	cubic centimetre
CMF	Consensus Market Forecast
CN_{FREE}	free cyanide
CNwad	weak-acid dissociable cyanide
CNDP	French National Public Debates
CoG	cut-off grade
Columbus	Columbus Gold Corporation
COTAM	Convention d'Occupation Temporaire du Domaine Privé de l'Etat pour activités minières
CPT	cone penetration testing
CSG	contribution sociale généralisée
Cwi	Bond low energy impact
CWP	contact water pond
CWR	coarse waste rock
CWRD	Central Waste Rock Dump
DB	Declining balance
DEAL	*Direction de l'environnement*, *de l'aménagement et du 304iameter304* (
DEM	digital elevation mapping
dia.	Diameter
EA	Environmental assessment
EDF-SI	Electricité de France – Systèmes Energétiques Insulaire
EDF	Électricité de France
EDS	Ensemble Detrique Superieur
EGRG	Extended Gravity Recoverable Gold
EIPPCB	European Commission's European Integrated Pollution Prevention and Control Bureau
EIS	Environmental Impact Statement
EMP	Environmental Management Plan
EoM	End of Mine
EPCM	Engineering, Procurement and Construction Management
ENE	east northeast
ESIA	Environmental, Social and Impact Assessment
ESHS	Environment, Social, Health and Safety
EU	European Union
EURUSD	Euro:USD exchange rate
FA	fire assay
FA-AAS	atomic absorption spectroscopy finish
FEL	front end loader
FCF	
FoS	Factor of Safety
~~ft~~	~~foot (feet)~~
g	gram
g/cm^3	grams per cubic centimetre
g/L	gram per litre
g/t	grams per tonne
gal	gallon
gpm	gallons per minute
GRG	Gravity Recoverable Gold

Abbreviation	Unit or Term
G&A	General and administrative
H:V	Horizontal to 1 vertical
Ha	hectares
HARD	half absolute relative difference
HDPE	High Density Polyethylene
HG	high-grade
Hg	Mercury
HS	Hazard study
HTW	horizontal true width
ICL	Intensive cyanide leach
ICP	induced couple plasma
ICPE	Facilities Classified for Environmental Protection
IED	
IDW^2	inverse distance weighting squared
IDW^3	inverse distance weighting cubed
IFC	International Finance Corporation
ILR	In line Leach Reactor
INRAP	Institut National de Recherches Archéologiques Préventives
IRR	Internal rate of return
k	kinetic constant
K	equilibrium constant
koz	thousand ounces
kg	kilograms
kg/m^3	kilograms per cubic metre
kg/t	kilograms per tonne
km	kilometre
km/h	kilometre per hour
km^2	square kilometre
kN/m^3	kilonewtons per cubic metre
KNAG	Kinetic Net Acid Generation
koz	thousand troy ounces
koz/y	thousand troy ounces per year
kPA	kiloPascals
kt	thousand tonnes
kt/d	thousand tonnes per day
kt/y	thousand tonnes per year
kV	kilovolt
kW	kilowatt
kWh	kilowatt-hour
kWh/t	kilowatt-hour per metric tonne
~~kx~~	~~Seismic coefficient~~
L	Litre
L/sec	litres per second
L/sec/m	litres per second per metre
La Source	La Source Développement
LAN	Local area network
lb	pound
LBM	London Bullion market
LFZ	Lower Favorable Zone
LG	low-grade
LL	Liquid Limit
LLDPE	Linear Low Density Polyethylene
LoM	Life-of-Mine
Lycopodium	Lycopodium Minerals Pty Ltd
m	metre
m.y.	million years

Abbreviation	Unit or Term
m/s	metres per second
m^2	square metre
m^3	cubic metre
m^3/d	cubic metres per day
MA	mechanical availability
Ma	Megaannum
MARN	Ministry of the Environment and Natural Resources
masl	metres above sea level
MEDDE	Ministry of l'écologie, du développement durable et de l'énergie
MG	medium-grade
mg/L	milligrams/litre
ML	clayey silt
mm	millimetre
mm^2	square millimetre
Mm^3	million cubic metre
Moz	million troy ounces
MPa	MEgaPascals
MSA	Mine services area
Mt	million tonnes
Mt/y	million tonnes per year
MW	million watts
NaCN	Sodium cyanide
NAG	non-acid generating
NE	northeast
NGO	non-governmental organization
Ni	nickel
NGT	North Guiana Trough
NI 43-101	Canadian National Instrument 43-101
NOL	Net Operating Losses
Nordgold	Nord Gold SE
NPR	neutralization potential ratio
NSR	Net Smelter Return
NPV	Net present value
NW	northwest
OMC	Orway Mineral Consultants
OMS	operations, maintenance and surveillance
ONF	*Office National des Forêts* French National Forestry Board
OSC	Ontario Securities Commission
oz	troy ounce
PAG	potentially acid generating
Pb	Lead
PEA	Preliminary Economic Assessment
PER	exclusive exploration permit
PEX	Exploitation permit
PFS	Prefeasibility Study
PGA	peak ground acceleration
PGB	Paramaca Greenstone Belt
PL	Plastic Limit
PLC	Programmable Logic Controller
PLS	Pregnant Leach Solution
PLU	Local Urbanism Plan
PMF	probable maximum flood
PMP	Probable Maximum Precipitation
Pocock	Pocock International
ppm	parts per million
P_{80}	80% passing

Abbreviation	Unit or Term
QA/QC	Quality Assurance/Quality Control
QEMSCAN	Quantitative Evaluation of Minerals by Scanning Electron
QP	Qualified Person
RBD	Managed Biological Reserve
RBI	Integral Biological Reserve
RBI LDD	Lucifer Dékou-Dékou biological reserves
RC	rotary circulation
RGF	Reseau Geodesique Francais
RI	
RoM	Run-of-Mine
RQD	Rock Quality Description
Ru	
R&R	Rest and relaxation
SAG	Semi-autogenous grinding
Sb	Antimony
sec	second
SDOM	Schéma Départemental d'Orientation Minière
SEEDs	Safety Evaluation of Existing Dams
SG	specific gravity
SGS	SGS Lakefield
SLM	Saint Laurent du Maroni
SMBS	sodium metabisulphite
SMC	SAG Mill Comminution
SMS	Semi-massive sulphide
SMU	Selective Mining Unit
SO_2	Sulfur Dioxide
SO_2 / g	Sulfur Dioxide per gram
SOTRAPMAG	SOTRAPMAG S.A.S.
SPI	SAG Power Index
SPT	standard penetration testing
SRCE	Standardized Reclamation Cost Estimator
SRK	SRK Consulting (U.S.), Inc.
st	short ton (2,000 pounds)
t	tonne (metric ton) (2,204.6 pounds)
t/d	tonnes per day
t/h	tonnes per hour
t/y	tonnes per year
Tanon	Tanon S.A.
TEM	Technical Economic Model
TGC	Terracognita Geological Consulting Inc.
TMS	Trace Mineral Search
TOS	trade off study
TSF	Tailings Storage Facility
TSP	total suspended particulates
TSS	total suspended solids.
TTG	tonalite, trondhjemite and granodiorite
UCS	Unconfined compressive strength
UFZ	Upper Felsic Zone
U-Pb	Uranium–lead
UQÀM	Université du Québec à Montréal
USCS	atomic absorption spectroscopy finish
UTM	Universal Transverse Mercator
V	Volts
VAT	value-added tax
VMS	volcanogenic massive sulphide
W	watt

Abbreviation	Unit or Term
WRD	waste rock dump
WTP	Water treatment plant
WWRD	West Waste Rock Dump
y	year
Zn	Zinc
US$/t	US$ per tonne

Appendices

Appendix A: Certificates of Qualified Persons

SRK Consulting (U.S.), Inc.
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CERTIFICATE OF QUALIFIED PERSON

I, Bart A. Stryhas, PhD, CPG, do hereby certify that:

1. I am Principal Resource Geologist of SRK Consulting (U.S.), Inc., 1125 Seventeenth Street, Suite 600, Denver, CO, USA, 80202.
2. This certificate applies to the technical report titled “NI 43-101 Technical Report, Bankable Feasibility Study, Montagne d’Or Project, French Guiana” with an Effective Date of March 6, 2017 (the “Technical Report”).
3. I graduated with a Doctorate degree in Structural Geology from Washington State University in 1988. In addition, I have obtained a Master of Science degree in Structural Geology from the University of Idaho in 1985 and a Bachelor of Arts degree in Geology from the University of Vermont in 1983. I am a current member of the American Institute of Professional Geologists. I have worked as a Geologist for a total of 30 years since my graduation from university. My relevant experience includes minerals exploration, mine geology, project development and resource estimation. I have conducted resource estimations since 1988 and have been involved in technical reports since 2004.
4. I have read the definition of “qualified person” set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purposes of NI 43-101.
5. I visited the Montagne d’Or property on April 1 through 3, 2014 and on October 12 through 17, 2015.
6. I am responsible for the preparation of background, geology and resource estimation Sections 1.1-1.4, 1.16.1, 4 except 4.3 and 4.4, 6-12, 14, 23, 25.1, 26.1 of the Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101
8. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement was in the preparation of the technical reports listed below:
    - “Amended NI 43-101 Technical Report, Preliminary Economic Assessment, Montagne d’Or Gold Deposit, Paul Isnard Project, Commune of Saint-Laurent-du-Maroni, NW French Guiana” with an Effective Date of July 8, 2015, prepared for Nord Gold N.V.;
    - “NI 43-101 Technical Report on Updated Resources, Montagne d’Or Gold Deposit, Paul Isnard Project, Commune of Saint-Laurent-du-Maroni, NW French Guiana” with an Effective Date of April 11, 2015, prepared for Nord Gold N.V. with Columbus Gold Corporation;
    - “Updated NI 43-101 Technical Report, Paul Isnard Project, French Guiana” with an Effective Date of February 1, 2012, prepared for Columbus Gold Corporation;
    - “Updated NI 43-101 Technical Report on Resources, Columbus Gold Corporation, Paul Isnard Project, French Guiana” with an Effective Date of January 13, 2011, prepared for Columbus Gold Corporation; and
    - “NI 43-101 Preliminary Assessment, Golden Star Resources Ltd., Paul Isnard Project, French Guiana” with an Effective Date of February 29, 2008, prepared for Golden Star Resources Ltd.;
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.

10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 28th Day of April, 2017.

“Signed” “Sealed”

Bart A. Stryhas, PhD, CPG

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CERTIFICATE OF QUALIFIED PERSON

I, John Tinucci, PhD, PE, do hereby certify that:

1. I am President and Principal Geotechnical Mining Engineer of SRK Consulting (U.S.), Inc., 1125 Seventeenth Street, Suite 600, Denver, CO, USA, 80202.
2. This certificate applies to the technical report titled “NI 43-101 Technical Report, Bankable Feasibility Study, Montagne d’Or Project, French Guiana” with an Effective Date of March 6, 2017 (the “Technical Report”).
3. I graduated with a degree in B.S. in Civil Engineering from Colorado State University, in 1980. In addition, I have obtained a M.S. in Geotechnical Engineering from University of California, Berkeley, in 1983 and I have obtained a Ph.D. in Geotechnical Engineering, Rock Mechanics from the University of California, Berkeley in 1985. I am member of the American Rock Mechanics Association, a member of the International Society of Rock Mechanics, a member of the ASCE GeoInstitute, and a Registered Member of the Society for Mining, Metallurgy & Exploration. I have worked as a Mining and Geotechnical Engineer for a total of 32 years since my graduation from university. My relevant experience includes 34 years of professional experience. I have 15 years managerial experience leading project teams, managing P&L operations for 120 staff, and directed own company of 8 staff for 8 years. I have technical experience in mine design, prefeasibility studies, feasibility studies, geomechanical assessments, rock mass characterization, project management, numerical analyses, underground mine stability, subsidence, tunneling, ground support, slope design and stabilization, excavation remediation, induced seismicity and dynamic ground motion. My industry commodities experience includes salt, potash, coal, platinum/palladium, iron, molybdenum, gold, silver, zinc, diamonds, and copper. My mine design experience includes open pit, room and pillar, (single and multi-level), conventional drill-and-blast and mechanized cutting, longwall, steep narrow vein, cut and fill, block caving, sublevel caving and cut and fill longhole stoping and paste backfilling.
4. I have read the definition of “qualified person” set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purposes of NI 43-101.
5. I visited the Montagne d’Or property on October 12 and 13, 2015
6. I am responsible for the preparation of geotechnical Sections 1.5, 1.16.2, 16.1, 25.2, 26.2 of the Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101
8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 28th Day of April, 2017.

“Signed” “Sealed”

John Tinucci, PhD, PE

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CERTIFICATE OF QUALIFIED PERSON

I, Bret Swanson, BEng Mining, MAusIMM, MMSAQP, do hereby certify that:

1. I am Principal Mining Engineer of SRK Consulting (U.S.), Inc., 1125 Seventeenth Street, Suite 600, Denver, CO, USA, 80202.
2. This certificate applies to the technical report titled "NI 43-101 Technical Report, Bankable Feasibility Study, Montagne d'Or Project, French Guiana" with an Effective Date of March 6, 2017 (the "Technical Report").
3. I graduated with a degree in Bachelor of Engineering in Mining Engineering from the University of Wollongong in 1995. I am a current member of the Mining & Metallurgical Society of America #01418QP. I have worked as a Mining Engineer for a total of 22 years since my graduation from university. My relevant experience includes contributions to numerous feasibility, pre-feasibility, preliminary assessment and competent person reports while employed with SRK, Denver. Previously, I worked on the design and implementation of mine planning and scheduling systems, long term mine design with environmental focus, and mine planning corporate standards for Solid Energy, New Zealand. In addition, I have worked in various sales and support roles utilizing Vulcan Software and MineSuite Production Statistics where I gained considerable exposure to mining operations and projects around the world.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I visited the Montagne d'Or property on October 12 and 13, 2015.
6. I am responsible for the preparation of mine design and mine planning Sections 1.6, 1.7.1, 1.16.3 (shared), 15, 16.2, 16.3, 25.3, 25.4 (shared), 26.3 of the Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101
8. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement was in the preparation of the technical reports listed below:
   - "Amended NI 43-101 Technical Report, Preliminary Economic Assessment, Montagne d'Or Gold Deposit, Paul Isnard Project, Commune of Saint-Laurent-du-Maroni, NW French Guiana" with an Effective Date of July 8, 2015, prepared for Nord Gold N.V.; and
   - "NI 43-101 Preliminary Assessment, Golden Star Resources Ltd., Paul Isnard Project, French Guiana" with an Effective Date of February 29, 2008, prepared for Golden Star Resources Ltd.;
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 28th Day of April, 2017.

"Signed" "Sealed"

Bret Swanson, BEng Mining, MAusIMM, MMSAQP

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CERTIFICATE OF QUALIFIED PERSON

I, Peter Clarke, BSc Mining, MBA, Peng, do hereby certify that:

1. I am Principal Consultant (Mining Engineer) of SRK Consulting (U.S.), Inc., 1125 Seventeenth Street, Suite 600, Denver, CO, USA, 80202.
2. This certificate applies to the technical report titled "NI 43-101 Technical Report, Bankable Feasibility Study, Montagne d'Or Project, French Guiana" with an Effective Date of March 6, 2017 (the "Technical Report").
3. I graduated with a degree in B.Sc. degree in Mining Engineering from University of Leeds in 1975. In addition, I have obtained an MBA granted by the University of Phoenix in 2002. I am a registered member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia since 1982. I have worked as a mining engineer for a total of 34 years since my graduation from university. My relevant includes experience as an open-pit mining engineer in mining operations and mine engineering consulting. Experience includes mining of precious metals, copper, lead, zinc, nickel, and industrial minerals in North America and overseas. I have an extensive background in open-pit mine design, planning, production scheduling, equipment selection and cost estimating. Studies conducted include property evaluations, scoping studies, feasibilities, mine planning optimizations, and due diligence.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I visited the Montagne d'Or property on October 16 and 17, 2015.
6. I am responsible for the preparation of mining Sections 1.7.2, 1.16.3 (shared), 2, 3, 5.1, 5.3, 5.5, 5.6 (shared), 5.7 (shared), 16.4, 25.4 (shared), 26.3 (shared), 26.12, 27 and 28.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101
8. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement was as a peer reviewer of the technical report titled "Amended NI 43-101 Technical Report, Preliminary Economic Assessment, Montagne d'Or Gold Deposit, Paul Isnard Project, Commune of Saint-Laurent-du-Maroni, NW French Guiana" with an Effective Date of July 8, 2015.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 28th Day of April, 2017.

"Signed" "Sealed"

Peter Clarke, BSc Mining, MBA, PEng

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CERTIFICATE OF QUALIFIED PERSON

I, Eric Olin, MSc Metallurgy, MBA, SME-RM, MAusIMM, do hereby certify that:

1. I am Principal Process Metallurgist of SRK Consulting (U.S.), Inc., 1125 Seventeenth Street, Suite 600, Denver, CO, USA, 80202.
2. This certificate applies to the technical report titled "NI 43-101 Technical Report, Bankable Feasibility Study, Montagne d'Or Project, French Guiana" with an Effective Date of March 6, 2017 (the "Technical Report").
3. I graduated with a Master of Science degree in Metallurgical Engineering from the Colorado School of Mines in 1976. I am a Registered Member of The Society for Mining, Metallurgy and Exploration, Inc. I have worked as a Metallurgist for a total of 39 years since my graduation from the Colorado School of Mines. My relevant experience includes extensive consulting, plant operations, process development, project management and research & development experience with base metals, precious metals, ferrous metals and industrial minerals. I have served as the plant superintendent for several gold and base metal mining operations. Additionally, I have been involved with numerous third-party due diligence audits, and preparation of project conceptual, pre-feasibility and full-feasibility studies.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I did not visit the Montagne d'Or property.
6. I am responsible for the preparation of mineral processing and metallurgy Sections 1.8, 1.16.4, 13, 25.5, 26.4 of the Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101
8. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement was in the preparation of the technical reports titled "NI 43-101 Technical Report on Updated Resources, Montagne d'Or Gold Deposit, Paul Isnard Project, Commune of Saint-Laurent-du-Maroni, NW French Guiana" with an Effective Date of April 11, 2015, and "Amended NI 43-101 Technical Report, Preliminary Economic Assessment, Montagne d'Or Gold Deposit, Paul Isnard Project, Commune of Saint-Laurent-du-Maroni, NW French Guiana" with an Effective Date of July 8, 2015.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 28th Day of April, 2017.

"Signed" "Sealed"

____

Eric Olin, MSc Metallurgy, MBA, SME-RM, MAusIMM

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CERTIFICATE OF QUALIFIED PERSON

I, David Bird, MSc, PG, SME-RM, do hereby certify that:

1. I am Principal Consultant (Geochemistry) of SRK Consulting (U.S.), Inc., 1125 Seventeenth Street, Suite 600, Denver, CO, USA, 80202.
2. This certificate applies to the technical report titled "NI 43-101 Technical Report, Bankable Feasibility Study, Montagne d'Or Project, French Guiana" with an Effective Date of March 6, 2017 (the "Technical Report").
1. I graduated with Bachelor's Degrees in Geology and Business Administration Management from Oregon State University in 1983. In addition, I obtained a Master's Degree in Geochemistry/Hydrogeology from the University of Nevada-Reno in 1993. I am a Registered Member of the Society for Mining, Metallurgy, and Exploration (SME). I am a certified Professional Geologist in the State of Oregon (G1438). I have worked full time as a Geologist and Geochemist for a total of 32 years. My relevant experience includes design, execution, and interpretation of mine waste geochemical characterization programs in support of open pit and underground mine planning and environmental impact assessments, design and supervision of water quality sampling and monitoring programs, geochemical modeling, and management of the geochemistry portion of numerous PEA, PFS and FS-level mine projects in the US and abroad.
2. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
3. I visited the Montagne d'Or property on October 12 and 13, 2015, and November 10 and 11, 2015.
4. I am responsible for the preparation of geochemical testing of tailings and waste rock, interpretation of data, and predictive geochemical modeling of tailings, waste rock, and pit lake Sections 1.12.3 and 26.8.2 of the Technical Report.
5. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101
6. I have not had prior involvement with the property that is the subject of the Technical Report.
7. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
8. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 28th Day of April, 2017.

"Signed" "Sealed"

David Bird, MSc, PG, SME-RM

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CERTIFICATE OF QUALIFIED PERSON

I, Paul Williams, MSc, PG, PH, SME-RM, do hereby certify that:

1. I am Principal Consultant (Hydrogeologist) of SRK Consulting (U.S.), Inc., 1125 Seventeenth Street, Suite 600, Denver, CO, USA, 80202.
2. This certificate applies to the technical report titled “NI 43-101 Technical Report, Bankable Feasibility Study, Montagne d’Or Project, French Guiana” with an Effective Date of March 6, 2017 (the “Technical Report”).
3. I graduated with a degree in Geological Engineering from Montana Tech in 1985. In addition, I obtained a Master’s Degree in Civil Engineering from University of Massachusetts at Lowell in 1993. I am a Registered Member of the Society for Mining, Metallurgy, and Exploration (SME). I am a certified Professional Geologist (Wyoming PG-2538), and a certified Professional Hydrogeologist (98-HG-1500) with the American Institute of Hydrology. I have full time worked as a Hydrogeologist for a total of 30 years.. My relevant experience includes design and implementation of large-scale field projects to support dewatering studies, groundwater flow modelling in support of open pit and underground mine planning and mine dewatering, and management of the hydrogeology portion of numerous PEA, PFS and FS-level mine projects in the US and abroad.
4. I have read the definition of “qualified person” set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purposes of NI 43-101.
5. I visited the Montagne d’Or property October 12 and 13, 2015, and June 28 and 29, 2016.
6. I am responsible for the preparation of hydrogeology Sections 1.12.1, 1.16.8 (shared) ,16.1.9 (shared), 16.5 (shared), 16.5.2, and 26.8.1 of the Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101
8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 28th Day of April, 2017.

“Signed” “Sealed”

Paul Williams, MSc, PG, PH, SME-RM

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CERTIFICATE OF QUALIFIED PERSON

I, David Hoekstra, BSc Civil Engineering, P.E. do hereby certify that:

1. I am Principal Consultant (Civil Engineer) of SRK Consulting (U.S.), Inc., 1125 Seventeenth Street, Suite 600, Denver, CO, USA, 80202.
2. This certificate applies to the technical report titled "NI 43-101 Technical Report, Bankable Feasibility Study, Montagne d'Or Project, French Guiana" with an Effective Date of March 6, 2017 (the "Technical Report").
3. I graduated with a degree in Civil Engineering from Colorado State University in 1986. I am a Professional Engineer of the States of Alaska, Colorado, Montana, South Carolina, and Wyoming. I have worked as an Engineer for a total of 30 years since my graduation from university. My relevant experience includes the design and implementation of mine water management systems and storm water controls for numerous PEA, PFS, FS-level and operating mine projects in the US and abroad.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I did not visit the Montagne d'Or property.
6. I am responsible for the preparation of hydrology Sections 1.12.2, 1.16.8 (shared), 16.1.9 (shared), 16.5 (shared), 18.2.3, 25.9, 26.8.1 of the Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101
8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 28th Day of April, 2017.

"Signed" "Sealed"

____

David Hoekstra, BSc Civil Engineering, P.E.

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CERTIFICATE OF QUALIFIED PERSON

I, Cam Scott, B.A.Sc., Geological Engineering, do hereby certify that:

1. I am Principal Geotechnical Engineer of SRK Consulting (Canada), Inc., 2200 – 1066 West Hastings Street, Vancouver BC, Canada V6E 3X2.
2. This certificate applies to the technical report titled "NI 43-101 Technical Report, Bankable Feasibility Study, Montagne d'Or Project, French Guiana" with an Effective Date of March 6, 2017 (the "Technical Report").
3. I graduated with B.Ap.Sc. degree in Geotechnical Engineering granted by the University of British Columbia in 1974 and an M.Eng. degree in Geotechnical Engineering granted by the University of Alberta in 1984. My relevant experience includes mine waste and mill site development. I have been a registered member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia since 1978. I have worked as a Geotechnical Engineer for a total of 42 years since my graduation from University of British Columbia in 1974.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I visited the Montagne d'Or property on October 16 and 17, 2015.
6. I am responsible for the preparation of TSF Sections 1.11, 1.16.7, 18.3, 25.8, 26.7 of the Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101
8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 28th Day of April, 2017.

"Signed" "Sealed"

Cam Scott, B.A.Sc., Geological Engineering

Lycopodium

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Australia
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CERTIFICATE OF QUALIFIED PERSON

I, David Gordon, B App Sc Engineering Metallurgy, FAusIMM, do hereby certify that:

1. I am Manager of Process of Lycopodium Minerals Pty Ltd, Level 5, 1 Adelaide Terrace, East Perth, Western Australia, 6004, Australia.
2. This certificate applies to the technical report titled "NI 43-101 Technical Report, Bankable Feasibility Study, Montagne d'Or Project, French Guiana" with an Effective Date of March 6, 2017 (the "Technical Report").
3. I graduated with a degree in Engineering Metallurgy from the Western Australian Institute of Technology in 1983. I am a Fellow of the AusIMM. I have worked as a Process Engineer for a total of 32 years since my graduation from university. My relevant experience includes over 15 years design experience encompassing all aspects of gold processing from testwork to process plant design and over 10 years experience in operating process plants.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I did not visit the Montagne d'Or property.
6. I am responsible for the preparation of process, recovery and infrastructure Sections 1.9, 1.10, 1.14 (shared), 1.16.5, 1.16.6, 5.2, 5.4, 5.6 (shared), 5.7 (shared), 17, 18.1, 18.2.1, 18.2.2, 21 (shared), 24, 25.6, 25.7, 26.5, and 26.6 of the Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 28th day of April, 2017.

David Gordon, B App Sc Engineering Metallurgy, FAusIMM

Celebrating 25 years of Engineering and Project Delivery Excellence

SRK Consulting (U.S.), Inc.
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CERTIFICATE OF QUALIFIED PERSON

I, Mark Willow, MSc, CEM, SME-RM, do hereby certify that:

1. I am Practice Leader of SRK Consulting (U.S.), Inc., 5250 Neil Road, Reno, Nevada 89511.
2. This certificate applies to the technical report titled "NI 43-101 Technical Report, Bankable Feasibility Study, Montagne d'Or Project, French Guiana" with an Effective Date of March 6, 2017 (the "Technical Report").
3. I graduated with Bachelor's degree in Fisheries and Wildlife Management from the University of Missouri in 1987 and a Master's degree in Environmental Science and Engineering from the Colorado School of Mines in 1995. I have worked as Biologist/Environmental Scientist for a total of 22 years since my graduation from university. I am a Registered Member of the Society for Mining, Metallurgy & Exploration (SME) No. 4104492 (expiration: Dec. 31, 2018). My relevant experience includes environmental due diligence/competent persons evaluations of developmental phase and operational phase mines through the world, including small gold mining projects in Panama, Senegal, Peru and Colombia; open pit and underground coal mines in Russia; several large copper mines and processing facilities in Mexico; and a mine/coking operation in China. My Project Manager experience includes several site characterization and mine closure projects. I work closely with the U.S. Forest Service and U.S. Bureau of Land Management on several permitting and mine closure projects to develop uniquely successful and cost effective closure alternatives for the abandoned mining operations. Finally, I draw upon this diverse background for knowledge and experience as a human health and ecological risk assessor with respect to potential environmental impacts associated with operating and closing mining properties, and have experienced in the development of Preliminary Remediation Goals and hazard/risk calculations for site remedial action plans under CERCLA activities according to current U.S. EPA risk assessment guidance. I am a Certified Environmental Manager (CEM) in the State of Nevada (#1832) in accordance with Nevada Administrative Code NAC 459.970 through 459.9729. Before any person consults for a fee in matters concerning: the management of hazardous waste; the investigation of a release or potential release of a hazardous substance; the sampling of any media to determine the release of a hazardous substance; the response to a release or cleanup of a hazardous substance; or the remediation soil or water contaminated with a hazardous substance, they must be certified by the Nevada Division of Environmental Protection, Bureau of Corrective Action;
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I visited the Montagne d'Or property on April 1 through 3, 2014
6. I am responsible for the preparation of environmental studies, permitting and social or community impact Sections 1.13, 1.16.9, 4.4, 20, 25.10, 26.9 of the Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101
8. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement was in the preparation of the technical report titled "Amended NI 43-101 Technical Report, Preliminary Economic Assessment, Montagne d'Or Gold Deposit, Paul Isnard Project, Commune of Saint-Laurent-du-Maroni, NW French Guiana" with an Effective Date of July 8, 2015.

9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 28th Day of April, 2017.

“Signed” “Sealed”

Mark Willow, MSc, CEM, SME-RM

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CERTIFICATE OF QUALIFIED PERSON

I, Grant Malensek, MEng, PEng/PGeo, do hereby certify that:

1. I am Principal Consultant (Mineral Project Evaluation) of SRK Consulting (U.S.), Inc., 1125 Seventeenth Street, Suite 600, Denver, CO, USA, 80202.
2. This certificate applies to the technical report titled "NI 43-101 Technical Report, Bankable Feasibility Study, Montagne d'Or Project, French Guiana" with an Effective Date of March 6, 2017 (the "Technical Report").
3. I graduated with a degree in B.S. Geological Sciences from University of British Columbia in 1987. In addition, I have obtained a M.E. in Geological Engineering (Colorado School of Mines, 1997). I am a Professional Engineer of the Association of Professional Engineers & Geoscientists of British Columbia. I have worked as an Engineer for a total of over 22 years since my graduation from university. My relevant experience includes business experience in financial analysis, project management and business development.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I did not visit the Montagne d'Or property.
6. I am responsible for preparations of the economics and market Sections 1.14 (shared), 1.15, 1.16.10, 1.16.11, 4.3, 19, 21 (shared), 22, 25.11, 25.12, 26.10 and 26.11 of the Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101
8. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement is in the preparation of the technical report titled, "Amended NI 43-101 Technical Report, Preliminary Economic Assessment, Montagne d'Or Gold Deposit, Paul Isnard Project, Commune of Saint-Laurent-du-Maroni, NW French Guiana" with an Effective Date of July 8, 2015, prepared for Nord Gold N.V.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 28th Day of April, 2017.

"Signed" "Sealed"

____

Grant Malensek, MEng, PEng/PGeo

Appendix B: Technical Economic Model

Economic Model Annual Summary

Company **Nordgold**
Project Name **Montagne d'Or**
Scenario Name **Base Case 4.5Mtpa - Opt 1**
Analysis Type **BFS - Base Case**

CALENDAR YEAR				**2019**	**2020**	**2021**	**2022**	**2023**	**2024**	**2025**	**2026**	**2027**	**2028**	**2029**	**2030**	**2031**	**2032**	**2033**	**2034**	**2035**
Project Timeline			**LoM**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**	**15**	**16**	**17**
Production Timeline			**Total**	**-3**	**-2**	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**
Discount Factors	EOP @ 5%	US$ & Metric Units	**or Avg.**	1.0000	0.9524	0.9070	0.8638	0.8227	0.7835	0.7462	0.7107	0.6768	0.6446	0.6139	0.5847	0.5568	0.5303	0.5051	0.4810	0.4581
Market Prices																				
Gold		US$/oz	**$1,250**	$1,250	$1,250	$1,250	$1,250	$1,250	$1,250	$1,250	$1,250	$1,250	$1,250	$1,250	$1,250	$1,250	$1,250	$1,250	$1,250	$1,250
Exchange Rate		EURUSD	**1.05**	1.05	1.05	1.05	1.05	1.05	1.05	1.05	1.05	1.05	1.05	1.05	1.05	1.05	1.05	1.05	1.05	1.05
Physicals							2,530	45,530.62	1.73											
Total Ore Mined		kt	54,114	-	173	235	5,231	5,470	5,673	5,135	5,713	6,071	4,364	4,924	5,508	5,335	281	-	-	-
Total Waste Mined		kt	241,753	-	3,650	3,660	17,161	25,613	28,860	29,957	21,931	26,560	21,000	24,300	20,857	17,508	696	-	-	-
Total Material Mined		kt	295,867	-	3,823	3,895	22,392	31,083	34,534	35,091	27,644	32,631	25,364	29,224	26,365	22,843	977	-	-	-
Strip Ratio		w/o	4.5	-	21.1	15.6	3.3	4.7	5.1	5.8	3.8	4.4	4.8	4.9	3.8	3.3	2.5	-	-	-
Total Ore Tonnes, Processed		kt	54,115	-	-	-	4,442	4,562	4,563	4,562	4,575	4,563	4,562	4,564	4,575	4,563	4,563	4,021	-	-
Gold Grade, Processed		g/t	1.58	-	-	-	1.63	1.91	1.87	1.95	1.80	1.45	1.76	1.44	1.77	1.71	0.83	0.73	-	-
Contained Gold, Processed		koz	2,745	-	-	-	233	280	274	286	265	212	258	211	260	251	121	94	-	-
Average Recovery, Gold		%	93.8%	--	--	--	94.2%	94.0%	94.0%	93.6%	93.6%	93.7%	93.6%	93.6%	93.6%	93.6%	93.6%	94.6%	--	--
Recovered Gold, Dore		koz	2,574	-	-	-	220	263	258	268	248	199	241	198	243	234	113	89	-	-
Payable Gold, Dore		koz	2,572	-	-	-	219	263	257	268	248	199	241	198	243	234	113	89	-	-
Cash Flow			1.45																	
Total Revenue		**$000s**	**3,214,654**	**-**	**-**	**-**	**274,172**	**328,844**	**321,604**	**334,650**	**309,385**	**248,251**	**301,532**	**247,055**	**303,994**	**292,695**	**141,478**	**110,993**	**-**	**-**
Mining Cost		$000s	(704,040)	-	-	-	(61,346)	(67,981)	(74,068)	(78,039)	(66,907)	(73,260)	(66,766)	(64,814)	(65,003)	(61,427)	(14,049)	(10,381)	-	-
Process Cost		$000s	(621,830)	-	-	-	(48,676)	(50,258)	(51,549)	(53,010)	(53,207)	(52,907)	(53,222)	(52,966)	(53,324)	(53,255)	(53,307)	(46,149)	-	-
Site G&A Cost		$000s	(224,309)	-	-	-	(19,036)	(19,147)	(19,220)	(19,294)	(18,963)	(19,183)	(18,999)	(18,926)	(18,926)	(18,778)	(17,328)	(16,509)	-	-
Water Management Cost		$000s	(6,368)	-	-	-	(250)	(375)	(433)	(419)	(637)	(606)	(629)	(659)	(697)	(891)	(402)	(104)	(56)	(51)
Refining/Selling Cost		$000s	(2,375)	-	-	-	(203)	(243)	(238)	(247)	(229)	(183)	(223)	(183)	(225)	(216)	(105)	(82)	-	-
Direct Cash Costs		**$000s**	**(1,558,922)**	**-**	**-**	**-**	**(129,511)**	**(138,003)**	**(145,507)**	**(151,009)**	**(139,942)**	**(146,139)**	**(139,839)**	**(137,547)**	**(138,174)**	**(134,567)**	**(85,191)**	**(73,225)**	**(56)**	**(51)**
Royalties		$000s	(153,374)	-	-	-	(13,676)	(16,403)	(16,042)	(16,692)	(15,432)	(12,383)	(15,040)	(12,323)	(13,092)	(11,967)	(5,785)	(4,538)	-	-
Indirect Cash Costs		**$000s**	**(153,374)**	**-**	**-**	**-**	**(13,676)**	**(16,403)**	**(16,042)**	**(16,692)**	**(15,432)**	**(12,383)**	**(15,040)**	**(12,323)**	**(13,092)**	**(11,967)**	**(5,785)**	**(4,538)**	**-**	**-**
Total Operating Expense		**$000s**	**(1,712,296)**	**-**	**-**	**-**	**(143,187)**	**(154,406)**	**(161,549)**	**(167,702)**	**(155,374)**	**(158,522)**	**(154,880)**	**(149,871)**	**(151,267)**	**(146,534)**	**(90,976)**	**(77,763)**	**(56)**	**(51)**
Operating Margin		**$000s**	**1,502,358**	**-**	**-**	**-**	**130,985**	**174,438**	**160,055**	**166,948**	**154,011**	**89,729**	**146,653**	**97,184**	**152,728**	**146,161**	**50,502**	**33,230**	**(56)**	**(51)**
Earnings Before Interest, Taxes & Depreciation		**$000s**	**1,502,358**	**-**	**-**	**-**	**130,985**	**174,438**	**160,055**	**166,948**	**154,011**	**89,729**	**146,653**	**97,184**	**152,728**	**146,161**	**50,502**	**33,230**	**(56)**	**(51)**
Depreciation Allowance		$000s	(766,131)	(8,826)	(9,582)	(20,197)	(83,057)	(72,645)	(61,722)	(60,288)	(49,905)	(48,113)	(49,877)	(43,868)	(37,757)	(34,399)	(31,939)	(153,954)	-	-
Other Non-Cash Tax Adjustments		$000s	-	-	-	-	5,489	4,684	7,325	4,476	4,422	15,526	(1,866)	4,252	(768)	2,838	(22,763)	(23,615)	-	-
Earnings Before Taxes		**$000s**	**736,228**	**(8,826)**	**(9,582)**	**(20,197)**	**53,417**	**106,477**	**105,658**	**111,137**	**108,529**	**57,142**	**94,910**	**57,568**	**114,202**	**114,600**	**(4,200)**	**(144,340)**	**(56)**	**(51)**
Income Tax		$000s	(200,746)	-	-	-	-	(27,002)	(29,104)	(17,439)	(29,794)	(3,708)	(12,770)	(15,033)	(32,777)	(33,121)	-	-	-	-
Net Income		**$000s**	**535,482**	**(8,826)**	**(9,582)**	**(20,197)**	**53,417**	**79,475**	**76,554**	**93,698**	**78,735**	**53,434**	**82,140**	**42,536**	**81,425**	**81,480**	**(4,200)**	**(144,340)**	**(56)**	**(51)**
Non-Cash Add Back - Depreciation		$000s	766,131	8,826	9,582	20,197	83,057	72,645	61,722	60,288	49,905	48,113	49,877	43,868	37,757	34,399	31,939	153,954	-	-
Other Non-Cash Tax Adjustments		$000s	-	-	-	-	(5,489)	(4,684)	(7,325)	(4,476)	(4,422)	(15,526)	1,866	(4,252)	768	(2,838)	22,763	23,615	-	-
Working Capital		$000s	0	-	-	-	(6,249)	(195)	270	(341)	1,060	353	(171)	192	(422)	45	2,313	1,483	1,661	-
Operating Cash Flow		**$000s**	**1,301,612**	**-**	**-**	**-**	**124,736**	**147,241**	**131,221**	**149,169**	**125,278**	**86,374**	**133,713**	**82,344**	**119,529**	**113,085**	**52,815**	**34,713**	**1,604**	**(51)**
Initial Capital		$000s	(535,191)	(143,255)	(204,818)	(187,118)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining Capital		$000s	(231,120)	-	-	-	(39,844)	(15,532)	(8,848)	(48,121)	(10,813)	(39,686)	(53,424)	(7,305)	(2,798)	(3,753)	(699)	(27)	(89)	-
Closure/Reclamation Capital		$000s	(60,659)	-	-	-	-	-	-	-	-	(9)	(480)	(1,129)	-	(254)	(1,227)	(4,538)	(11,257)	(13,241)
Total Capital		**$000s**	**(826,971)**	**(143,255)**	**(204,818)**	**(187,118)**	**(39,844)**	**(15,532)**	**(8,848)**	**(48,121)**	**(10,813)**	**(39,695)**	**(53,904)**	**(8,434)**	**(2,798)**	**(4,007)**	**(1,926)**	**(4,565)**	**(11,346)**	**(13,241)**
Surplus Tax Credit Refunds		**$000s**	**185,632**	-	48,854	65,836	59,691	11,250	-	-	-	-	-	-	-	-	-	-	-	-
Metrics																				
Economic Metrics																				
a) Pre-Tax																				
Free Cash Flow		$000s	**861,019**	(143,255)	(155,964)	(121,282)	144,583	169,960	151,478	118,487	144,258	50,388	92,578	88,942	149,508	142,199	50,889	30,148	(9,742)	(13,292)
Cumulative Free Cash Flow		$000s		(143,255)	(299,219)	(420,501)	(275,918)	(105,957)	45,520	164,007	308,266	358,653	451,232	540,174	689,682	831,881	882,770	912,918	903,176	889,885
NPV @ 5%		$000s	**506,731**	(143,255)	(148,537)	(110,007)	124,897	139,827	118,687	88,417	102,522	34,104	59,677	54,603	87,414	79,182	26,988	15,227	(4,686)	(6,089)
Cumulative NPV		$000s		(143,255)	(291,792)	(401,799)	(276,902)	(137,075)	(18,389)	70,028	172,550	206,654	266,331	320,934	408,349	487,530	514,518	529,744	525,059	518,969
IRR		%	**22.2%**																	
Undiscounted Payback From Start of Comm. Prod.		Years	**3.7**			-	-	-	3.7	3.7	3.7	3.7	3.7	3.7	3.7	3.7	3.7	3.7	3.7	3.7
PI @ 5%		NPV / (PW of TC)	**0.72**	143,255	195,065	169,722	34,419	12,778	6,932	35,908	7,685	26,867	34,747	5,178	1,636	2,231	1,021	2,306	5,458	6,066
b) After-Tax																				
Free Cash Flow		$000s	**660,273**	(143,255)	(155,964)	(121,282)	144,583	142,959	122,374	101,049	114,464	46,680	79,809	73,910	116,731	109,078	50,889	30,148	(9,742)	(13,292)
Cumulative Free Cash Flow		$000s		(143,255)	(299,219)	(420,501)	(275,918)	(132,959)	(10,585)	90,463	204,928	251,607	331,416	405,326	522,057	631,135	682,025	712,172	702,431	689,139
NPV @ 5%		$000s	**369,949**	(143,255)	(148,537)	(110,007)	124,897	117,613	95,883	75,404	81,348	31,595	51,445	45,374	68,250	60,739	26,988	15,227	(4,686)	(6,089)
Cumulative NPV		$000s		(143,255)	(291,792)	(401,799)	(276,902)	(159,289)	(63,407)	11,997	93,345	124,940	176,385	221,759	290,010	350,749	377,736	392,963	388,277	382,188
IRR		%	**18.7%**																	
Undiscounted Payback from Start of Comm. Prod.		Years	**4.1**			-	-	-	-	4.1	4.1	4.1	4.1	4.1	4.1	4.1	4.1	4.1	4.1	4.1
PI @ 5%		NPV / (PW of TC)	**0.53**	143,255	195,065	169,722	34,419	12,778	6,932	35,908	7,685	26,867	34,747	5,178	1,636	2,231	1,021	2,306	5,458	6,066
Operating Metrics																				
Mine Life		Years	**12**																	
Average Mining Rate (Mill Feed + Waste)		kt/y	**35,091**																	
Average Processing Rate		kt/y	**4,575**																	
Mining Cost		$ / t mined	**$2.44**	-	-	-	2.74	2.19	2.14	2.22	2.42	2.25	2.63	2.22	2.47	2.69	14.38	-	-	-
Mining Cost		$ / t ore	**$13.01**	-	-	-	11.73	12.43	13.06	15.20	11.71	12.07	15.30	13.16	11.80	11.51	49.98	-	-	-
Processing Cost		$ / t ore	**$11.49**	-	-	-	10.96	11.02	11.30	11.62	11.63	11.60	11.67	11.61	11.65	11.67	11.68	11.48	-	-
G&A Cost	17%	$ / t ore	**$4.15**	-	-	-	4.29	4.20	4.21	4.23	4.15	4.20	4.16	4.15	4.14	4.12	3.80	4.11	-	-
Water Mgmt Cost		$ / t ore	**$0.12**	-	-	-	0.06	0.08	0.09	0.09	0.14	0.13	0.14	0.14	0.15	0.20	0.09	0.03	-	-
Sales Metrics																				
LoM Au Dore Sales		koz	**2,572**	-	-	-	219	263	257	268	248	199	241	198	243	234	113	89	-	-
LoM Annual Dore Sales		koz / yr	**214**																	
LoM All-In Sustaining Costs (AISC)		$ / oz	**$779**	-	-	-	834	646	662	806	671	998	866	801	633	643	821	927	-	-
Total Au Dore Sales (1st 10 Yrs)		koz	**2,370**	-	-	-	219	263	257	268	248	199	241	198	243	234	-	-	-	-
Avg. Annual Dore Sales (1st Ten Yrs)		koz / yr	**237**																	
All-In Sustaining Costs (AISC - 1st Ten Yrs)		$ / oz	**$749**																	
Average Grade (1st Ten Yrs)		g/t Au	**1.73**																	

Appendix C: Tax Models

COMPANY	Nordgold
BUSINESS UNIT	Montagne d'Or
OPERATION	BFS - Base Case

Year	value / factor	units	Total or Avg.	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035
Project Counter				1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
Production Timeline				-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14
Time Before Mine Closure				15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	-1	-2
PRODUCTION SUMMARY																				
Mill Feed	-	kt	**54,113,619**	-	-	-	4,441,875	4,562,500	4,562,500	4,562,500	4,575,000	4,562,500	4,562,500	4,562,500	4,575,000	4,562,500	4,562,500	4,021,744	-	-
Recovered Gold	-	oz	**2,574,317**	-	-	-	219,567	263,352	257,539	267,992	247,763	198,795	241,455	197,850	243,436	234,390	113,296	88,883	-	-
INCOME TAXES (BY YEAR)																				
Corporate Income Tax																				
Gross Revenue		$000s	**3,214,654**	-	-	-	274,172	328,844	321,604	334,650	309,385	248,251	301,532	247,055	303,994	292,695	141,478	110,993	-	-
Less Refining/Selling Costs		$000s	**(2,375)**	-	-	-	(203)	(243)	(238)	(247)	(229)	(183)	(223)	(183)	(225)	(216)	(105)	(82)	-	-
Less Mining Royalty		$000s	**(153,374)**	-	-	-	(13,676)	(16,403)	(16,042)	(16,692)	(15,432)	(12,383)	(15,040)	(12,323)	(13,092)	(11,967)	(5,785)	(4,538)	-	-
Less Operating Costs		$000s	**(1,556,547)**	-	-	-	(129,309)	(137,760)	(145,270)	(150,762)	(139,713)	(145,956)	(139,616)	(137,365)	(137,950)	(134,350)	(85,087)	(73,143)	(56)	(51)
Less Stockpile Adjustments		$000s	**-**	-	-	-	5,489	4,684	7,325	4,476	4,422	15,526	(1,866)	4,252	(768)	2,838	(22,763)	(23,615)	-	-
Adjusted EBITDA		**$000s**	**1,502,358**	**-**	**-**	**-**	**136,474**	**179,122**	**167,380**	**171,424**	**158,433**	**105,255**	**144,787**	**101,436**	**151,960**	**148,999**	**27,739**	**9,615**	**(56)**	**(51)**
Standard Depreciation		$000s	**(119,174)**	-	(684)	(1,617)	(6,432)	(6,335)	(6,540)	(7,015)	(6,981)	(6,758)	(8,131)	(7,591)	(7,049)	(6,562)	(5,963)	(41,515)	-	-
Accelerated Depreciation		$000s	**(646,957)**	(8,826)	(8,897)	(18,580)	(76,626)	(66,311)	(55,182)	(53,273)	(42,924)	(41,354)	(41,746)	(36,277)	(30,708)	(27,837)	(25,976)	(112,439)	-	-
EBT Before LCF		**$000s**	**736,228**	**(8,826)**	**(9,582)**	**(20,197)**	**53,417**	**106,477**	**105,658**	**111,137**	**108,529**	**57,142**	**94,910**	**57,568**	**114,202**	**114,600**	**(4,200)**	**(144,340)**	**(56)**	**(51)**
LCF Utilized	$1,050	$000s	**(47,632)**	-	-	-	(47,632)	-	-	-	-	-	-	-	-	-	-	-	-	-
Income Subject to Income Tax		**$000s**	**688,595**	**(8,826)**	**(9,582)**	**(20,197)**	**5,785**	**106,477**	**105,658**	**111,137**	**108,529**	**57,142**	**94,910**	**57,568**	**114,202**	**114,600**	**(4,200)**	**(144,340)**	**(56)**	**(51)**
Standard Corporate Income Tax Rate	28.0%	$000s	**245,282**	-	-	-	1,620	29,814	29,584	31,118	30,388	16,000	26,575	16,119	31,977	32,088	-	-	-	-
3.3% Surcharge Threshold on CIT	$801	$000s	**7,830**	-	-	-	27	957	950	1,000	976	502	851	505	1,029	1,032	-	-		
Total Income Tax Before FG Tax Credits		$000s	**253,112**	-	-	-	1,647	30,771	30,534	32,119	31,364	16,501	27,425	16,625	33,005	33,121	-	-	-	-
FG Tax Credits	LoM	$000s	**(52,366)**	-	-	-	(1,647)	(3,769)	(1,430)	(14,680)	(1,571)	(12,793)	(14,656)	(1,592)	(228)	-	-	-	-	-
Total Income Tax After Tax Credits	**29.2%**	**$000s**	**200,746**	**-**	**-**	**-**	**-**	**27,002**	**29,104**	**17,439**	**29,794**	**3,708**	**12,770**	**15,033**	**32,777**	**33,121**	**-**	**-**	**-**	**-**
Surplus Tax Credit Refund Payable		**$000s**	**185,724**	**48,854**	**65,836**	**59,691**	**11,250**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**74**	**9**	**9**	**-**
Loss Carry Forward																				
Beginning Balance	$9,027	$000s		(9,027)	(17,853)	(27,435)	(47,632)	-	-	-	-	-	-	-	-	-	-	(4,200)	(148,540)	(148,596)
Additions		$000s	**(187,412)**	(8,826)	(9,582)	(20,197)	-	-	-	-	-	-	-	-	-	-	(4,200)	(144,340)	(56)	(51)
Utilized		$000s	**47,632**	-	-	-	47,632	-	-	-	-	-	-	-	-	-	-	-	-	-
Ending Balance		$000s	**-**	(17,853)	(27,435)	(47,632)	-	-	-	-	-	-	-	-	-	-	(4,200)	(148,540)	(148,596)	(148,647)
FRENCH OVERSEAS DEPARTMENT TAX CREDITS																				
a) Moveable Assets																				
Major Mobile Equipment		$000s	**82,252**	22,058	4,817	33,403	5,355	2,610	3,436	2,321	2,257	583	2,489	2,398	524	-	-	-	-	-
Other Equipment		$000s	**7,727**	-	809	2,058	4,859	-	-	-	-	-	-	-	-	-	-	-	-	-
Subtotal Moveable Assets Capex		$000s	**89,979**	22,058	5,627	35,461	10,214	2,610	3,436	2,321	2,257	583	2,489	2,398	524	-	-	-	-	-
Eligible Tax Credit	35.0%	$000s	**31,492**	7,720	1,969	12,411	3,575	914	1,202	812	790	204	871	839	184	-	-	-	-	-
b) Buildings Assets																				
Process/Infrastructure: Construction Indirects		$000s	**46,131**	13,839	18,452	13,839	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Processing: Treatment Plant - All		$000s	**105,912**	31,774	42,365	31,774	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Processing: Reagents and Plant Services - All		$000s	**31,230**	9,369	12,492	9,369	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Infrastructure		$000s	**144,675**	43,402	57,870	43,402	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mining (Facilities) - All		$000s	**5,819**	1,746	2,327	1,746	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Management Costs - All		$000s	**28,473**	8,542	11,389	8,542	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Owners Project Costs		$000s	**29,515**	8,855	11,806	8,855	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Subtotal Eligible Buildings Asset Capex		$000s	**391,754**	117,526	156,702	117,526	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Eligible Tax Credit	35.0%	$000s	**137,114**	41,134	54,846	41,134	-	-	-	-	-	-	-	-	-	-	-	-	-	-
c) Other Assets																				
Preproduction (Pre-Stripping)		$000s	**33,461**	-	16,217	17,244	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Infrastructure (Sustaining)		$000s	**-**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
TSF (Sustaining) - TSF Lifts		$000s	**151,282**	-	-	-	26,422	7,186	-	39,574	1,243	35,843	39,384	1,631	-	-	-	-	-	-
Mine Water Management		$000s	**9,956**	-	8,213	50	24	237	650	47	129	127	-	232	35	-	210	-	-	-
Surface Water Management		$000s	**3,827**	-	1,345	266	189	735	-	-	859	-	-	287	93	-	-	27	27	-
Closure/Reclamation/Salvage/Post Closure Monitoring		$000s	**-**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Subtotal Other Assets Capex		$000s	**198,526**	-	25,775	17,559	26,634	8,159	650	39,622	2,231	35,970	39,384	2,150	128	-	210	27	27	-
Eligible Tax Credit	35.0%	$000s	**69,484**	-	9,021	6,146	9,322	2,856	228	13,868	781	12,589	13,784	753	45	-	74	9	9	-
Tax Credit Summary																				
Opening Balance	$0	$000s	**-**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Tax Credits Calculated (Incl "Other Assets" Credits?)	Yes	$000s	**238,090**	48,854	65,836	59,691	12,897	3,769	1,430	14,680	1,571	12,793	14,656	1,592	228	-	74	9	9	-
1) Tax Credits Utilized Against Income Taxes		$000s	**(52,366)**	-	-	-	(1,647)	(3,769)	(1,430)	(14,680)	(1,571)	(12,793)	(14,656)	(1,592)	(228)	-	-	-	-	-
Ending Balance Before Refund		$000s	**185,724**	48,854	65,836	59,691	11,250	-	-	-	-	-	-	-	-	-	74	9	9	-
2) Tax Credit Refunds in excess of CIT		$000s	**(185,724)**	(48,854)	(65,836)	(59,691)	(11,250)	-	-	-	-	-	-	-	-	-	(74)	(9)	(9)	-
Ending Balance After Refund		$000s	**-**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total LoM Eligible Assets Capex		**$000s**	**680,258**	139,584	188,103	170,547	36,848	10,770	4,086	41,943	4,488	36,553	41,873	4,548	652	-	210	27	27	-
Total LoM Tax Credits Generated		**$000s**	**238,090**	48,854	65,836	59,691	12,897	3,769	1,430	14,680	1,571	12,793	14,656	1,592	228	-	74	9	9	-
% of Tax Credits vs. Eligible LoM Project Capital		**%**	**35%**																	

COMPANY	Nordgold																			
BUSINESS UNIT	Montagne d'Or																			
OPERATION	BFS - Base Case																			
Year				**2019**	**2020**	**2021**	**2022**	**2023**	**2024**	**2025**	**2026**	**2027**	**2028**	**2029**	**2030**	**2031**	**2032**	**2033**	**2034**	**2035**
Project Counter				**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**	**15**	**16**	**17**
Production Timeline	**value /**		**Total**	**-3**	**-2**	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**
Time Before Mine Closure	**factor**	**units**	**or Avg.**	**15**	**14**	**13**	**12**	**11**	**10**	**9**	**8**	**7**	**6**	**5**	**4**	**3**	**2**	**1**	**-1**	**-2**
PRODUCTION SUMMARY																				
DEPRECIATION																				
Depreciable Capex Base																				
a) Straight line Capex																				
i) Equipment/Machinery (5 Yr)																				
Mining - Mobile Equipment (Rebuilds)		$000s	**32,781**	-	-	628	774	2,378	3,797	5,347	3,275	2,303	10,056	1,862	1,315	1,045	-	-	-	-
Mining - Other Equipment		$000s	**15,222**	-	3,422	4,719	7,081	-	-	-	-	-	-	-	-	-	-	-	-	-
Subtotal Equipment/Machinery (SL)		**$000s**	**48,003**	**-**	**3,422**	**5,347**	**7,855**	**2,378**	**3,797**	**5,347**	**3,275**	**2,303**	**10,056**	**1,862**	**1,315**	**1,045**	**-**	**-**	**-**	**-**
ii) Facilities/Buildings (20 yr)																				
Infrastructure - General		$000s	**35,527**	10,658	14,211	10,658	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Infrastructure - Environmental		$000s	**135**	41	54	41	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Infrastructure: Buildings - Admin & Security		$000s	**675**	202	270	202	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Infrastructure: Buildings - Plant		$000s	**12,183**	3,655	4,873	3,655	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Infrastructure - Permanent Accommodation		$000s	**17,005**	5,101	6,802	5,101	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mining - Facilities		$000s	**5,819**	1,746	2,327	1,746	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Subtotal Facilities/Buildings (SL)		**$000s**	**71,343**	**21,403**	**28,537**	**21,403**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
b) Declining Balance Capex																				
i) Equipment/Machinery (5 Yr)																				
Mining - Mobile Equipment (Initial & Repl.)		$000s	**82,252**	22,058	4,817	33,403	5,355	2,610	3,436	2,321	2,257	583	2,489	2,398	524	-	-	-	-	-
Owners Project Costs - Plant Mobile Equipment		$000s	**10,533**	3,160	4,213	3,160	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Infrastructure (Sustaining) - FEL/Bus/LV Repl.		$000s	**11,581**	-	-	-	-	489	831	831	2,981	831	831	831	831	2,639	489	-	-	-
Subtotal Equipment/Machinery (DB)		**$000s**	**104,365**	**25,217**	**9,030**	**36,563**	**5,355**	**3,099**	**4,266**	**3,152**	**5,237**	**1,414**	**3,320**	**3,229**	**1,355**	**2,639**	**489**	**-**	**-**	**-**
ii) Facilities/Buildings (15 Yr)																				
Capitalized Preproduction Costs		$000s	**52,003**	-	25,148	26,855	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Process/Infrastructure: Construction Indirects - All		$000s	**46,828**	14,048	18,731	14,048	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Processing: Treatment Plant - All		$000s	**105,912**	31,774	42,365	31,774	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Processing: Reagents and Plant Services - All		$000s	**31,230**	9,369	12,492	9,369	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Infrastructure - Water & Sewerage		$000s	**4,165**	1,250	1,666	1,250	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Infrastructure - Power Supply		$000s	**54,172**	16,251	21,669	16,251	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Infrastructure - Tailings Dam		$000s	**32,354**	9,706	12,942	9,706	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Management Costs - All		$000s	**28,473**	8,542	11,389	8,542	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Owners Project Costs - General		$000s	**18,983**	5,695	7,593	5,695	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Infrastructure (Sustaining) - WTP		$000s	**1,896**	-	-	-	-	1,896	-	-	-	-	-	-	-	-	-	-	-	-
TSF (Sustaining) - TSF Lifts		$000s	**151,282**	-	-	-	26,422	7,186	-	39,574	1,243	35,843	39,384	1,631	-	-	-	-	-	-
Water Management - All		$000s	**15,304**	-	9,834	316	213	973	785	47	1,058	127	664	583	128	69	210	27	89	-
Subtotal Facilities/Buildings (DB)		**$000s**	**542,600**	**96,635**	**163,828**	**123,805**	**26,634**	**10,055**	**785**	**39,622**	**2,301**	**35,970**	**40,048**	**2,213**	**128**	**69**	**210**	**27**	**89**	**-**
Depreciable Capex Base Summary																				
Capitalized Preproduction Costs		$000s	**52,003**	-	25,148	26,855	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mining		$000s	**130,255**	22,058	8,240	38,750	13,210	4,989	7,232	7,668	5,532	2,886	12,546	4,261	1,839	1,045	-	-	-	-
Process/Infrastructure		$000s	**403,991**	121,197	161,597	121,197	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Infrastructure (Sustaining)		$000s	**13,477**	-	-	-	-	2,385	831	831	2,981	831	831	831	831	2,639	489	-	-	-
TSF (Sustaining)		$000s	**151,282**	-	-	-	26,422	7,186	-	39,574	1,243	35,843	39,384	1,631	-	-	-	-	-	-
Water Management		$000s	**15,304**	-	9,834	316	213	973	785	47	1,058	127	664	583	128	69	210	27	89	-
Total Depreciable Capex		**$000s**	**766,312**	**143,255**	**204,818**	**187,118**	**39,844**	**15,532**	**8,848**	**48,121**	**10,813**	**39,686**	**53,424**	**7,305**	**2,798**	**3,753**	**699**	**27**	**89**	**-**
			84%																	
Check Calc.																				
E1 Total LoM Capex		$000s	**826,971**	143,255	204,818	187,118	39,844	15,532	8,848	48,121	10,813	39,695	53,904	8,434	2,798	4,007	1,926	4,565	11,346	13,241
Less E1 Non-Depreciable Closure Costs		$000s	**(60,659)**	-	-	-	-	-	-	-	-	(9)	(480)	(1,129)	-	(254)	(1,227)	(4,538)	(11,257)	(13,241)
Total E1 Depreciable Capex		$000s	**766,312**	143,255	204,818	187,118	39,844	15,532	8,848	48,121	10,813	39,686	53,424	7,305	2,798	3,753	699	27	89	-
Check		-	**ok**	ok	ok	ok	ok	ok	ok	ok	ok	ok	ok	ok	ok	ok	ok	ok	ok	ok
Standard Depreciation (SL) Calculations																				
a) Equipment/Machinery																				
Total Equipment/Machinery Capital		**$000s**	**48,003**	**-**	**3,422**	**5,347**	**7,855**	**2,378**	**3,797**	**5,347**	**3,275**	**2,303**	**10,056**	**1,862**	**1,315**	**1,045**	**-**	**-**	**-**	**-**
Opening Balance		$000s	-	-	-	2,738	6,468	11,459	11,070	11,893	13,792	13,654	12,765	18,257	16,096	13,928	11,979	9,583	7,666	6,133
Additions		$000s	**48,003**	-	3,422	5,347	7,855	2,378	3,797	5,347	3,275	2,303	10,056	1,862	1,315	1,045	-	-	-	-
5 Yr SL Dep. When Acquired	20.0%		**47,831**	-	684	1,617	2,865	2,767	2,973	3,448	3,413	3,191	4,564	4,024	3,482	2,995	2,396	1,917	1,533	1,227
Closing Balance		$000s	-	-	2,738	6,468	11,459	11,070	11,893	13,792	13,654	12,765	18,257	16,096	13,928	11,979	9,583	7,666	6,133	4,907
Total Depreciation Taken		**$000s**	**47,831**	**-**	**684**	**1,617**	**2,865**	**2,767**	**2,973**	**3,448**	**3,413**	**3,191**	**4,564**	**4,024**	**3,482**	**2,995**	**2,396**	**1,917**	**1,533**	**1,227**
Depreciation Taken with EOM Write-off		**$000s**	**47,831**	**-**	**684**	**1,617**	**2,865**	**2,767**	**2,973**	**3,448**	**3,413**	**3,191**	**4,564**	**4,024**	**3,482**	**2,995**	**2,396**	**9,410**	**-**	**-**
b) Facilities/Buildings																				
20 Yr SL Dep. When Placed in Service	5.0%																			

COMPANY: Nordgold
BUSINESS UNIT: Montagne d'Or
OPERATION: BFS - Base Case

	value / factor	units	Total or Avg.	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035
Year				**2019**	**2020**	**2021**	**2022**	**2023**	**2024**	**2025**	**2026**	**2027**	**2028**	**2029**	**2030**	**2031**	**2032**	**2033**	**2034**	**2035**
Project Counter				**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**	**15**	**16**	**17**
Production Timeline				**-3**	**-2**	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**
Time Before Mine Closure				**15**	**14**	**13**	**12**	**11**	**10**	**9**	**8**	**7**	**6**	**5**	**4**	**3**	**2**	**1**	**-1**	**-2**
PRODUCTION SUMMARY																				
Total Facilities/Buildings Capital		**$000s**	**71,343**	**21,403**	**28,537**	**21,403**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
-3	-	%	-				5%	5%	5%	5%	5%	5%	5%	5%	5%	5%	5%	5%	5%	5%
-2	-	%	-				5%	5%	5%	5%	5%	5%	5%	5%	5%	5%	5%	5%	5%	5%
-1	-	%	-				5%	5%	5%	5%	5%	5%	5%	5%	5%	5%	5%	5%	5%	5%
1	-	%	-				5%	5%	5%	5%	5%	5%	5%	5%	5%	5%	5%	5%	5%	5%
2	-	%						5%	5%	5%	5%	5%	5%	5%	5%	5%	5%	5%	5%	5%
3	-	%	-						5%	5%	5%	5%	5%	5%	5%	5%	5%	5%	5%	5%
4	-	%	-							5%	5%	5%	5%	5%	5%	5%	5%	5%	5%	5%
5	-	%	-								5%	5%	5%	5%	5%	5%	5%	5%	5%	5%
6	-	%	-									5%	5%	5%	5%	5%	5%	5%	5%	5%
7	-	%	-										5%	5%	5%	5%	5%	5%	5%	5%
8	-	%	-											5%	5%	5%	5%	5%	5%	5%
9	-	%	-												5%	5%	5%	5%	5%	5%
10	-	%	-													5%	5%	5%	5%	5%
11	-	%	-														5%	5%	5%	5%
12	-	%	-															5%	5%	5%
13	-	%	-																5%	5%
14	-	%	-																	5%
15	-	%	-																	
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27	-	%	-																	
28	-	%	-																	
29	-	%	-																	
Standard Depreciation Calculator																				
-3	21,403	$000s	**21,403**	-	-	-	1,070	1,070	1,070	1,070	1,070	1,070	1,070	1,070	1,070	1,070	1,070	1,070	1,070	1,070
-2	28,537	$000s	**28,537**	-	-	-	1,427	1,427	1,427	1,427	1,427	1,427	1,427	1,427	1,427	1,427	1,427	1,427	1,427	1,427
-1	21,403	$000s	**21,403**	-	-	-	1,070	1,070	1,070	1,070	1,070	1,070	1,070	1,070	1,070	1,070	1,070	1,070	1,070	1,070
1	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
2	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
3	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
4	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
5	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
6	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
7	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
8	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
9	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
10	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
11	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
12	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
13	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
14	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
15	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
16	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
17	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
18	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
19	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
20	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
21	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
22	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
23	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
24	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
25	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
26	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
27	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
28	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
29	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Calc. Std Facilities/Bldgs Depreciation	**71,343**	**$000s**	**71,343**	**-**	**-**	**-**	**3,567**	**3,567**	**3,567**	**3,567**	**3,567**	**3,567**	**3,567**	**3,567**	**3,567**	**3,567**	**3,567**	**3,567**	**3,567**	**3,567**
Std Depreciation Utilized with EOM Writeoff		**$000s**	**71,343**	**-**	**-**	**-**	**3,567**	**3,567**	**3,567**	**3,567**	**3,567**	**3,567**	**3,567**	**3,567**	**3,567**	**3,567**	**3,567**	**32,104**	**-**	**-**

COMPANY	Nordgold
BUSINESS UNIT	Montagne d'Or
OPERATION	BFS - Base Case

Year				2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035
Project Counter				1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
Production Timeline	value /		Total	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14
Time Before Mine Closure	factor	units	or Avg.	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	-1	-2
PRODUCTION SUMMARY																				
Accelerated Depreciation (DB) Rates																				
a) Equipment/Machinery																				
Declining Balance Method	DBSL																			
DB Depreciation Modifier	1.75																			
Depreciation Life (in yrs) when acquired	5																			
Total Capital		**$000s**	**104,365**	**25,217**	**9,030**	**36,563**	**5,355**	**3,099**	**4,266**	**3,152**	**5,237**	**1,414**	**3,320**	**3,229**	**1,355**	**2,639**	**489**	**-**	**-**	**-**
-3		%	-	35.0%	22.8%	14.8%	13.7%	13.7%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%
-2		%	-		35.0%	22.8%	14.8%	13.7%	13.7%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%
-1		%	-			35.0%	22.8%	14.8%	13.7%	13.7%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%
1		%	-				35.0%	22.8%	14.8%	13.7%	13.7%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%
2		%						35.0%	22.8%	14.8%	13.7%	13.7%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%
3		%	-						35.0%	22.8%	14.8%	13.7%	13.7%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%
4		%	-							35.0%	22.8%	14.8%	13.7%	13.7%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%
5		%	-								35.0%	22.8%	14.8%	13.7%	13.7%	0.0%	0.0%	0.0%	0.0%	0.0%
6		%	-									35.0%	22.8%	14.8%	13.7%	13.7%	0.0%	0.0%	0.0%	0.0%
7		%	-										35.0%	22.8%	14.8%	13.7%	13.7%	0.0%	0.0%	0.0%
8		%	-											35.0%	22.8%	14.8%	13.7%	13.7%	0.0%	0.0%
9		%	-												35.0%	22.8%	14.8%	13.7%	13.7%	0.0%
10		%	-													35.0%	22.8%	14.8%	13.7%	13.7%
11		%	-														35.0%	22.8%	14.8%	13.7%
12		%	-															35.0%	22.8%	14.8%
13		%	-																35.0%	22.8%
14		%	-																	35.0%
15		%	-																	
16		%	-																	
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29		%	-																	
Accelerated Depreciation Calculator																				
-3	25,217	$000s	**25,217**	8,826	5,737	3,729	3,463	3,463	-	-	-	-	-	-	-	-	-	-	-	-
-2	9,030	$000s	**9,030**	-	3,161	2,054	1,335	1,240	1,240	-	-	-	-	-	-	-	-	-	-	-
-1	36,563	$000s	**36,563**	-	-	12,797	8,318	5,407	5,021	5,021	-	-	-	-	-	-	-	-	-	-
1	5,355	$000s	**5,355**	-	-	-	1,874	1,218	792	735	735	-	-	-	-	-	-	-	-	-
2	3,099	$000s	**3,099**	-	-	-	-	1,085	705	458	426	426	-	-	-	-	-	-	-	-
3	4,266	$000s	**4,266**	-	-	-	-	-	1,493	971	631	586	586	-	-	-	-	-	-	-
4	3,152	$000s	**3,152**	-	-	-	-	-	-	1,103	717	466	433	433	-	-	-	-	-	-
5	5,237	$000s	**5,237**	-	-	-	-	-	-	-	1,833	1,191	774	719	719	-	-	-	-	-
6	1,414	$000s	**1,414**	-	-	-	-	-	-	-	-	495	322	209	194	194	-	-	-	-
7	3,320	$000s	**3,320**	-	-	-	-	-	-	-	-	-	1,162	755	491	456	456	-	-	-
8	3,229	$000s	**3,229**	-	-	-	-	-	-	-	-	-	-	1,130	735	477	443	443	-	-
9	1,355	$000s	**1,355**	-	-	-	-	-	-	-	-	-	-	-	474	308	200	186	186	-
10	2,639	$000s	**2,639**	-	-	-	-	-	-	-	-	-	-	-	-	924	600	390	362	362
11	489	$000s	**489**	-	-	-	-	-	-	-	-	-	-	-	-	-	171	111	72	67
12	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
13	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
14	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
15	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
16	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
17	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
18	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
19	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
20	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
21	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
22	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
23	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
24	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
25	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-

COMPANY	Nordgold
BUSINESS UNIT	Montagne d'Or
OPERATION	BFS - Base Case

Year				2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035
Project Counter				1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
Production Timeline	value /		Total	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14
Time Before Mine Closure	factor	units	or Avg.	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	-1	-2
PRODUCTION SUMMARY																				
26	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
27	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
28	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
29	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Calculated Depreciation	**104,365**	**$000s**	**104,365**	**8,826**	**8,897**	**18,580**	**14,990**	**12,412**	**9,251**	**8,288**	**4,342**	**3,164**	**3,277**	**3,246**	**2,613**	**2,359**	**1,871**	**1,131**	**621**	**429**
Accel. Depr. Utilized with EOM Writeoff		**$000s**	**104,365**	**8,826**	**8,897**	**18,580**	**14,990**	**12,412**	**9,251**	**8,288**	**4,342**	**3,164**	**3,277**	**3,246**	**2,613**	**2,359**	**1,871**	**2,248**	**-**	**-**

b) Facilities/Buildings

Declining Balance Method	DBSL																			
DB Depreciation Modifier	2.25																			
Depreciation Life (in yrs) when placed in service	15																			
Total Capital		**$000s**	**542,600**	**96,635**	**163,828**	**123,805**	**26,634**	**10,055**	**785**	**39,622**	**2,301**	**35,970**	**40,048**	**2,213**	**128**	**69**	**210**	**27**	**89**	**-**
-3		%	**-**				15.0%	12.8%	10.8%	9.2%	7.8%	6.7%	5.7%	4.8%	4.1%	3.9%	3.9%	3.9%	3.9%	3.9%
-2		%	**-**				15.0%	12.8%	10.8%	9.2%	7.8%	6.7%	5.7%	4.8%	4.1%	3.9%	3.9%	3.9%	3.9%	3.9%
-1		%	**-**				15.0%	12.8%	10.8%	9.2%	7.8%	6.7%	5.7%	4.8%	4.1%	3.9%	3.9%	3.9%	3.9%	3.9%
1		%	**-**				15.0%	12.8%	10.8%	9.2%	7.8%	6.7%	5.7%	4.8%	4.1%	3.9%	3.9%	3.9%	3.9%	3.9%
2		%	**-**					15.0%	12.8%	10.8%	9.2%	7.8%	6.7%	5.7%	4.8%	4.1%	3.9%	3.9%	3.9%	3.9%
3		%	**-**						15.0%	12.8%	10.8%	9.2%	7.8%	6.7%	5.7%	4.8%	4.1%	3.9%	3.9%	3.9%
4		%	**-**							15.0%	12.8%	10.8%	9.2%	7.8%	6.7%	5.7%	4.8%	4.1%	3.9%	3.9%
5		%	**-**								15.0%	12.8%	10.8%	9.2%	7.8%	6.7%	5.7%	4.8%	4.1%	3.9%
6		%	**-**									15.0%	12.8%	10.8%	9.2%	7.8%	6.7%	5.7%	4.8%	4.1%
7		%	**-**										15.0%	12.8%	10.8%	9.2%	7.8%	6.7%	5.7%	4.8%
8		%	**-**											15.0%	12.8%	10.8%	9.2%	7.8%	6.7%	5.7%
9		%	**-**												15.0%	12.8%	10.8%	9.2%	7.8%	6.7%
10		%	**-**													15.0%	12.8%	10.8%	9.2%	7.8%
11		%	**-**														15.0%	12.8%	10.8%	9.2%
12		%	**-**															15.0%	12.8%	10.8%
13		%	**-**																15.0%	12.8%
14		%	**-**																	15.0%
15		%	**-**																	
16		%	**-**																	
17		%	**-**																	
18		%	**-**																	
19		%	**-**																	
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28		%	**-**																	
29		%	**-**																	

Accelerated Depreciation Calculator

-3	96,635	$000s	**96,635**	-	-	-	14,495	12,321	10,473	8,902	7,567	6,432	5,467	4,647	3,950	3,730	3,730	3,730	3,730	3,730
-2	163,828	$000s	**163,828**	-	-	-	24,574	20,888	17,755	15,092	12,828	10,904	9,268	7,878	6,696	6,324	6,324	6,324	6,324	6,324
-1	123,805	$000s	**123,805**	-	-	-	18,571	15,785	13,417	11,405	9,694	8,240	7,004	5,953	5,060	4,779	4,779	4,779	4,779	4,779
1	26,634	$000s	**26,634**	-	-	-	3,995	3,396	2,887	2,454	2,085	1,773	1,507	1,281	1,089	1,028	1,028	1,028	1,028	1,028
2	10,055	$000s	**10,055**	-	-	-	-	1,508	1,282	1,090	926	787	669	569	484	411	388	388	388	388
3	785	$000s	**785**	-	-	-	-	-	118	100	85	72	61	52	44	38	32	30	30	30
4	39,622	$000s	**39,622**	-	-	-	-	-	-	5,943	5,052	4,294	3,650	3,102	2,637	2,241	1,905	1,619	1,530	1,530
5	2,301	$000s	**2,301**	-	-	-	-	-	-	-	345	293	249	212	180	153	130	111	94	89
6	35,970	$000s	**35,970**	-	-	-	-	-	-	-	-	5,395	4,586	3,898	3,313	2,816	2,394	2,035	1,730	1,470
7	40,048	$000s	**40,048**	-	-	-	-	-	-	-	-	-	6,007	5,106	4,340	3,689	3,136	2,665	2,266	1,926
8	2,213	$000s	**2,213**	-	-	-	-	-	-	-	-	-	-	332	282	240	204	173	147	125
9	128	$000s	**128**	-	-	-	-	-	-	-	-	-	-	-	19	16	14	12	10	9
10	69	$000s	**69**	-	-	-	-	-	-	-	-	-	-	-	-	10	9	7	6	5
11	210	$000s	**210**	-	-	-	-	-	-	-	-	-	-	-	-	-	32	27	23	19
12	27	$000s	**27**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	4	3	3
13	89	$000s	**89**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	13	11
14	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
15	69	$000s	**69**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
16	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
17	113	$000s	**105**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
18	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
19	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
20	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
21	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-

COMPANY	Nordgold
BUSINESS UNIT	Montagne d'Or
OPERATION	BFS - Base Case

Year				2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035
Project Counter				1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
Production Timeline	value /		Total	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14
Time Before Mine Closure	factor	units	or Avg.	15	14	13	12	11	10	9	8	7	6	5	4	3	2	1	-1	-2
PRODUCTION SUMMARY																				
22	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
23	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
24	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
25	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
26	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
27	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
28	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
29	0	$000s	**0**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Calculated Depreciation	**542,600**	**$000s**	**542,591**	**-**	**-**	**-**	**61,635**	**53,898**	**45,931**	**44,985**	**38,582**	**38,190**	**38,469**	**33,031**	**28,095**	**25,477**	**24,105**	**22,934**	**22,102**	**21,467**
Accel. Depr. Utilized with EOM Writeoff		**$000s**	**542,591**	**-**	**-**	**-**	**61,635**	**53,898**	**45,931**	**44,985**	**38,582**	**38,190**	**38,469**	**33,031**	**28,095**	**25,477**	**24,105**	**110,191**	**-**	**-**

	Declining Balance w/ EOP Writeoff			
	DB Modifier	125	175	225
	Asset Life	3-4 Yrs	5-6 Yrs	>6 Yrs
Year	5	10	15	20
1	0.3500	0.2250	0.1500	0.1125
2	0.2275	0.1744	0.1275	0.0998
3	0.1479	0.1351	0.1084	0.0886
4	0.0961	0.1047	0.0921	0.0786
5	0.1785	0.0812	0.0783	0.0698
6	0.0000	0.0629	0.0666	0.0619
7	0.0000	0.0488	0.0566	0.0550
8	0.0000	0.0378	0.0481	0.0488
9	0.0000	0.0293	0.0409	0.0433
10	0.0000	0.1009	0.0347	0.0384
11	0.0000	0.0000	0.0295	0.0341
12	0.0000	0.0000	0.0251	0.0303
13	0.0000	0.0000	0.0213	0.0269
14	0.0000	0.0000	0.0181	0.0238
15	0.0000	0.0000	0.1028	0.0212
16	0.0000	0.0000	0.0000	0.0188
17	0.0000	0.0000	0.0000	0.0167
18	0.0000	0.0000	0.0000	0.0148
19	0.0000	0.0000	0.0000	0.0131
20	0.0000	0.0000	0.0000	0.1036
Total	**1.0000**	**1.0000**	**1.0000**	**1.0000**

	Declining Balance Switching to SL			
	DB Modifier	125	175	225
	Asset Life	3-4 Yrs	5-6 Yrs	>6 Yrs
Year	5	10	15	20
1	0.3500	0.2250	0.1500	0.1125
2	0.2275	0.1744	0.1275	0.0998
3	0.1479	0.1351	0.1084	0.0886
4	0.1373	0.1047	0.0921	0.0786
5	0.1373	0.0812	0.0783	0.0698
6	0.0000	0.0629	0.0666	0.0619
7	0.0000	0.0542	0.0566	0.0550
8	0.0000	0.0542	0.0481	0.0488
9	0.0000	0.0542	0.0409	0.0433
10	0.0000	0.0542	0.0386	0.0384
11	0.0000	0.0000	0.0386	0.0341
12	0.0000	0.0000	0.0386	0.0303
13	0.0000	0.0000	0.0386	0.0298
14	0.0000	0.0000	0.0386	0.0298
15	0.0000	0.0000	0.0386	0.0298
16	0.0000	0.0000	0.0000	0.0298
17	0.0000	0.0000	0.0000	0.0298
18	0.0000	0.0000	0.0000	0.0298
19	0.0000	0.0000	0.0000	0.0298
20	0.0000	0.0000	0.0000	0.0298
Total	**1.0000**	**1.0000**	**1.0000**	**1.0000**